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Control of Particulate Emissions in the Primary Nonferrous Metals Industries

Symposium Proceedings

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The purpose of the symposium on "Control of Particulate Emissions in the Primary Nonferrous Metals Industries" was to provide a forum for the exchange of knowledge and new ideas on particulate control technology with emphasis on industrial applications of environmental particulate control technology in the primary nonferrous industries. The symposium held at Monterey, California, March 18-21, 1979 was sponsored by the U. S. Environmental Protection Agency, Office of Research and Development, Metals and Inorganic Chemicals Branch, Industrial Environmental Research Laboratory-Cincinnati.

The symposium included presentations on foreign and domestic technology applicable to copper, lead, zinc, and aluminum as well as discussions on other advanced technology, measurement techniques, and current EPA programs that are pertinent to particulate control in the nonferrous metals industry.

Speakers from England, Canada, Sweden, Japan, Australia, and the United States discussed recent developments and technology for particulate control in nonferrous operations.

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FOREWORD

When energy and material resources are extracted, processed, converted, and used, the related pollutional impacts on our environment and even on our health often require that new and increasingly efficient pollution control methods be used. The Industrial Environmental Research Laboratory - Cincinnati (IERL-Ci) assists in developing and demonstrating new and improved methodologies that will meet these needs both efficiently and economically.

To perform this duty effectively, the IERL-Ci coordinates and interacts with industry through conferences, symposia, and cooperative projects. This particular report is the proceedings of a speciality conference designed to provide a forum for the exchange of knowledge and new ideas on particulate control technology with emphasis on industrial applications of environmental particulate control technology in the primary nonferrous industries. The symposium, held at Monterey, California, March 18-21, 1979 was sponsored by the U. S. Environmental Protection Agency, Office of Research and Development, Metals and Inorganic Chemicals Branch, Industrial Environmental Research Laboratory-Cincinnati.

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Director
Industrial Environmental Research Laboratory
Cincinnati

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Birmingham, Alabama

EPA WELCOME AND INTRODUCTION TO THE SYMPOSIUM

George S. Thompson, Jr.
Chief
Metals and Inorganic Chemicals Branch
Industrial Environmental Research Laboratory
Cincinnati, Ohio

Welcome and Acknowledgements

I would like to extend to you EPA's "welcome" to this symposium on particulate control practices and problems in the nonferrous metals production industries. I offer a special thanks to Dick Meek and his colleagues of the Southern Research Institute for their aid in developing and administering the symposium, to the Session Chairmen, Bob Jaffee, Jerry Cole, Ivor Campbell, and Seymour Epstein, to the authors of the presentations to follow, to our Monday Dinner Speaker, Dr. William Dresher and the panel members for the discussion on Tuesday evening, and to the audience, those who have come to learn and to share their experiences and insights. We hope our efforts are successful in providing you with an understanding and an awareness of the objectives and status of EPA's R & D activities impacting the nonferrous metals production industry.

We solicit your participation and comments after each presentation and particularly at the Tuesday evening discussion session. Members of my staff who are involved with the nonferrous metals production R & D program include John Burckle and Fred Craig. They will be actively soliciting your comments and reactions during this conference's discussion periods and through informal contacts.

The Role of Research and Development

I represent EPA's Office of Research and Development, specifically the Industrial Environmental Research Laboratory in Cincinnati, Ohio. My Branch, the Metals and Inorganic Chemicals Branch, is responsible for conducting R & D activities on air, water, and solid waste pollution for a variety of industries, including the nonferrous metals industry, the inorganic chemicals industry, the metal finishing and fabrication industry, and a number of miscellaneous industries such as glass, cement, and asbestos. My Branch, as well as my Director's Division and the Laboratory in Cincinnati, came into existence three years ago as a result of a major reorganization within EPA's Office of Research and Development. The major benefit from this reorganization was the establishment of a new charter directing one research group to address pollution control RD & D for all three of the major environmental media--air, water, and land--for specific industries. Prior to the 1975 reorganization, the programs for RD & D involving pollution in the different media were conducted by different groups under different managment. Often, they did not coordinate their work. In most cases, these groups of Federal researchers were physically located in different parts of the United States. This reorganization has been instrumental in correcting these problems and bringing a cohesive approach to our control technology programs.

Our broad charter is suited extremely well to conducting valuable research programs; we are one of a small handful of EPA activities that can address the "total pollution problem." Also as a result of our broad charter, we have the capability to interface with other EPA offices and Federal Agencies having regulatory and enforcement responsibilities impacting the industrial sector. Allow me to provide a specific example that directly addresses your interests. My staff interfaces with the regulatory offices, such as the Effluent Guidelines Division, the Office of Solid Waste, the Office of Air Quality Planning and Standards, and EPA's newly structured Office of Toxic Substances. We interface with EPA's air and water enforcement offices as well as many of EPA's ten Regional Offices. What does this interface provide? Awareness--a basic requirement for conducting valuable research--and technical support--a "must" for establishing firm technical foundations for regulatory and enforcement actions. I must stress the following point: EPA's Office of Research and Development is an independent function within EPA; it does not report through line management to any one EPA regulatory or enforcement activity. If you're asking yourself "what does all of this mean?"--let me summarize. Our interface with these programs along with our interface with you, the industry, provides us with the awareness to structure our research activities to be best "in tune" with the most important needs. We in research can develop and implement programs that provide answers to key technical and economical pollution problems impacting the industry.

EPA's Nonferrous Metals Research Program

The nonferrous metals research program is structured around the following goals: (which will be described in detail by John Burckle)

- Advance the state-of-the-art in air, water, and solid waste pollution control and treatment technology.

- Provide EPA with the best technological basis for the setting and enforcing of regulations.

- Ensure both EPA and the industry that the abatement of pollution from one media will not result in either pollution to another media or excessive energy consumption.

- Act as a focal point for information dissemination.

These goals are not easily achieved. Everyone must work toward them. We in the Office of Research and Development must have a clear understanding of the industrial processes and the resulting air, water, and solid waste pollutants generated by these processes; we must be perceptive to both the short and long-term research needs within EPA and obviously outside of EPA. We rely strongly on ideas, direction toward problems, and expert advice from people like yourselves.

In conclusion, this symposium provides an excellent forum for us to exchange ideas and viewpoints. Please comment on our research program. While we are knowledgeable, we are not experts. However, you are faced with the "real world" problems that provide you with insight and practical working knowledge. If you share your insight and expertise to assist us in developing better programs, we will all benefit.

EPA NONFERROUS RESEARCH

Session Chairman: Robert I. Jaffee
Electric Power Research Institute
Palo Alto, California

EPA's NONFERROUS METALS PRODUCTION RESEARCH, DEVELOPMENT, AND DEMONSTRATION PROGRAM

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ABSTRACT

The U.S. Environmental Protection Agency is primarily a regulatory agency whose mission it is to attain and maintain environmental quality for the protection of human health and welfare. As a complement to its regulatory activities, the Agency supports research and development leading to improved pollution control technology. The Office of Research and Development is responsible for the development and management of a National program of research, development, and demonstration leading to an understanding of the origin and effects of pollution, techniques for the measurement and monitoring of environmental quality, and technology for the control, management, and reutilization of pollutants and wastes. The ORD's Nonferrous Metals Program sponsors research to provide technology for their solution in the nonferrous metals industry. This program is the responsibility of the Industrial Environmental Research Laboratory at Cincinnati. The Resource Extraction Branch is responsible for the ore extraction and beneficiation operations, and the Metals and Inorganic Chemicals Branch is responsible for the production and fabrication operations. This paper presents an overview of the research program dealing with the production of nonferrous metals.

INTRODUCTION

The EPA's nonferrous metals program sponsors research, development, and demonstration projects to define environmental problems caused by the mining and winning of nonferrous metals and to provide technology for their solution. This program is located in the Industrial Environmental Research Laboratory at Cincinnati. The Resource Extraction Branch is responsible for ore extraction and beneficiation aspects, and the Metals and Inorganic Chemicals Branch is responsible for the production (smelting and refining) and fabrication (tube, sheet, wire, etc.) operations. This paper presents an overview of the program dealing with the production of nonferrous metals.

The program is structured into two major components. The first component is problem identification and definition. Problem identification and definition is accomplished through a program of engineering analysis of process inputs and operations coupled with sampling and analyses of process inputs and waste streams to identify and characterize waste streams having a potentially serious effect upon the environment. A multimedia approach is utilized and energy and economic factors are taken into consideration in comparison of alternative process and control technologies.

The second phase of the program, the control technology RD & D phase, deals with the research, development and demonstration of feasible solutions. As a first step, the technical and economic feasibility of selected conceptual approaches are studied to identify the most promising candidates for support. The transfer of technology developed and applied in other areas or in foreign countries is the fundamental approach utilized.

PROBLEM IDENTIFICATION AND DEFINITION

Upon reassignment to Cincinnati in 1976, the program undertook as a first step the assessment and definition of the environmental problems arising from the production of nonferrous metals. This step, has resulted in the development of a data base containing a profile of the composition of the industry process information with associated air, water and solid waste emission data.

To date, the assessment and problem definition has been completed for all primary and secondary nonferrous metals production processes used in the United States. However, the assessment is currently based on information existing in the literature only. This task has resulted in a five volume report in various stages of review.

1. Overview
2. Primary Copper, Lead, and Zinc
3. Primary Aluminum
4. Primary Minor Nonferrous Metals
5. Secondary Nonferrous Metals

The first three volumes of this report have been distributed within the Agency and the selected outside organizations for review and comment. We are also preparing an update to survey newly developed information for future incorporation into the report.

In brief, the assessment shows the following. The nonferrous metals production industry of the United States is diverse and complex in its composition, product mix and environmental problems. It is responsible for the production of 35 commercially significant primary metals and the recovery of some 18 nonferrous metals from waste materials.

The primary industry is composed of about 200 mines and 200 plants involved in the various phases of winning metals from the ores. In order of commercial significance by tonnage production, the industry produces aluminum, copper, lead, zinc, and the minor metals. Sulfuric acid is a principal by-product. Depending on the product and ore composition, a number of processing steps are involved: mining, concentrating, metal winning, and refining into a high purity form through a large number of processes. These activities may be integrated into a single site, or they may be widespread geographically. Most domestic ores, excepting aluminum, are obtained from 75 large mines producing copper, lead and zinc ores; some additional 13 minor metals are obtained as coproducts from these ores. A number of these operations are located in highly urbanized areas, while a significant number are situated in remote, relatively unpopulated areas. Many of the plants are old, with only a small portion having been built since 1960.

The secondary industry collects scrap materials, and, through pyrometallurgical or chemical processes, recovers specific nonferrous metals for recycling into commerce. It is the most diffused segment being composed of 400 plants operated by 300 firms. These plants are generally small with 85% having less than 100 employees and 35% having less than 4 employees. They tend to be concentrated in the urban, highly industrial areas (East Coast, Great Lakes, and clusters on the West Coast).

Production of nonferrous metals in the United States generates great quantities of pollutants that enter the air, water, and soil. Many of the emissions having the potential for adverse impacts are controlled, but others are not. Some could be controlled by application of currently available technology; others will require development of new control techniques and systems.

The most important needs for improved environmental control are as follows:

1. Technology for more effective control of air emissions containing sulfur oxides and significant quantities of toxic metal compounds.

2. Methods for effective disposal of solid and sludge residues containing leachable toxic components.

3. More effective methods for treatment of acidic waters contaminated with toxic metal compounds.

4. More effective control methods for fugitive emissions containing SO_2 and toxic metal compounds.

5. Better definition of specific emission streams suspected to be hazardous, especially in plants producing the minor nonferrous metals in the secondary nonferrous metals industries.

6. Better definition of the potential impact of new technologies that may be used in the future for production of nonferrous metals.

Based upon what we learned from this first step, we have incorporated a continuing assessment effort into our program. This industry assessment is designed to provide information on the production technologies, the process emissions, the control technologies, and the environmental pollution problems of the nonferrous metals production industry. The assessment activity is composed of activities which lead to a highly specific quantitative multimedia analysis and problem definition of an industry segment. Information is identified, acquired, and subjected to engineering analysis in order to identify specific environmental pollution problems and potential technologies to solve these problems. The results of these efforts provide the specific program tactics for out-year efforts and redirection of ongoing projects as appropriate.

The assessment program is now composed of a number of projects which culminate in an assessment report to be updated annually. First, current information in the worldwide literature is screened on a bimonthly basis; the result of this screening is a publication entitled "Technical Awareness Bulletin - Nonferrous Metals." This Bulletin serves several useful functions: (1) it is distributed to interested Agency personnel to furnish up-to-date information on industry developments; (2) it is a source tool for the update of the assessment activities; (3) it is distributed to industry to inform and aid in technology transfer which will hopefully lead to improved environmental control.

The Nonferrous Metals Production Resource Center is the focal point of the assessment activities. The Center is being set up to acquire information (spotlighted by the Technical Awareness Bulletin, pertinent government reports, agency files, and newly developed data from field test programs) and perform an engineering analysis of selected situations to pinpoint environmental problems and develop conceptual solutions. This activity results in the annual update of the Environmental Assessment reports which form the basis for the development of specific program tactics for out-year planning. In addition, the report will specifically identify key pollution problems.

Specific effort is made to acquire information from foreign countries regarding advanced technologies and practices for add-on control devices or improved process technologies (Analysis of Foreign Technology). This project also furnishes the capability to review and analyze the technical results of our PL-480 programs. The scope of assessment activities is currently being expanded to provide for capabilities for analysis of

- existing technology, both add-on controls (retrofit or new plant) and (new) demonstrated process technology such as the Noranda and Mitshubishi systems
- potential research and development initiatives involving technology not yet applied to nonferrous metals production
- field testing

CONTROL TECHNOLOGY R & D

The control technology RD & D program covers the development of equipment and process modifications to control the multimedia environmental impacts resulting from the production of nonferrous metals to acceptable levels.

The control technology development program addresses 5 subareas: (1) weak SO_2 gas stream control, (2) particulate control with emphasis on volatile and toxic trace elements, (3) control of fugitive emissions, (4) solid waste management techniques, and (5) treatment techniques for metal-bearing acidic wastewaters. The program is structured to deal with these areas identified by the environmental assessment as the most important targets for improved control. Efforts programmed for the support of regulatory and enforcement programs include a demonstration of a flue gas desulfurization technology, preparatory work for demonstrations of control technologies for fugitives and other weak SO_2 gas stream controls, and research leading to the development of more effective air pollution control, water pollution treatment and solid waste containment systems.

- Air Pollution Problems

In recent years, the Agency has focused most of its regulatory and enforcement activities for this industry toward the abatement of SO_2 and total particulate emissions from primary smelters. These problems remain persistent and are still to be resolved. Besides these criteria pollutants, this industry has the potential to emit large tonnage quantities of airborne particulate composed of such metals as arsenic, cadmium, lead, and copper. Control devices, which are conventionally operated for recovery of metal values, a prevalent industry practice, have thus far been found to be ineffective collectors of the volatile form of these metals. Fugitive emissions containing arsenic, lead and possibly cadmium are also a significant source of toxic metals to the atmosphere. In the case of both processes and fugitive emissions, the emissions contain appreciable SO_2 and, therefore, the selection of control options must consider how control of all pollutants can best be accomplished.

Add-on control requires the development of more cost effective approaches to flue gas cleaning, especially where retrofit is concerned. The objectives of this work area are: (1) to identify the existence of more cost-effective add-on control technology and, through applications research, transfer existing technology from other fields into the nonferrous area; (2) to develop and document the performance of existing and new process technology having superior environmental and energy-conserving attributes coupled with competitive economics.

Weak SO_2 Control

Studies evaluating the technical feasibility of applying controls and process modifications to specific unit operations and gas blending have been completed. These studies clearly show that approaches such as oxygen enrichment, flue gas blending, and various FGD methods can, from a theoretical standpoint, be applied to metallurgical offgases to abate discharges of sulfur dioxide. However, no one technology is applicable to all situations, with the possible exception of FGD. Control technologies based upon process modifications, such as oxygen enrichment and gas stream blending, are dependent upon specific process conditions. The above studies were extended to investigate the technical and economic feasibility of maximizing total smelter SO_2 control to 95% or better of the smelter input sulfur. This study revealed that certain "advanced" systems held promise in being economically feasible for greenfield smelters.

We are going forward with our FY 78 initiative to develop and demonstrate a suitable Flue Gas Cleaning (FGC) technology for control of particulate and weak SO_2 from a molybdenum roaster. The above studies will serve a vital role in defining systems to be piloted. The technical and economic conclusions drawn from this demonstration along with the results of the above efforts will provide the definitive data to settle the question of control for weak SO_2 gas streams for primary copper smelters and support the Agency's NSO program. This demonstration is designed to provide for effective control of volatile and other trace metals contained in the weak SO_2 offgases; thus, an additional control alternative would be available for toxic metals.

Particulate Control

The program for the control of particulate emissions is being focused on the identification of technology which will provide acceptable environmental levels of control for volatile and toxic trace elements as well as gross particulate. The basic strategy is to (1) locate and test full-scale systems which are exemplary in control of trace, as well as gross, particulates, including volatiles; (2) apply new technologies developed by IERL-RTP on smelter off-gas slip streams as Demonstrations at pilot scale.

A baghouse controlling particulate emissions from a moly roaster has been tested for emissions control performance and as a clean-up device prior to SO_2 control. Fabric filtration will also be experimentally investigated as a means for control of particulate from a lead sinter machine and blast furnace in Yugoslavia. An exemplary fabric filter/hot ESP system for arsenic trioxide control with metals value recovery has been tested.

The performance of electrostatic precipitators for control of both gross particulate emissions and toxic, including volatile, trace elements is being investigated (ESP Evaluation). The first year's effort identified specific problems in the lack of control of arsenic and other volatile trace elements by electrostatic precipitation at elevated stack gas temperatures. Plans are also being formulated to conduct (including volatile and trace metal) studies on existing exemplary full-scale and promising advanced technology pilot systems.

Wet scrubbers for particulate control are applied primarily in the primary industry in the cleanup of acid plant feed gases and in certain segments of the secondary industry. They are also applied in the primary aluminum segment, but the preferred control today is "dry scrubbing." Our efforts in scrubber development are limited to: (1) applications testing of advanced technology-based systems as developed by IERL-RTP which offer significant cost advantages; (2) a scrubbing system which will permit simultaneous control of SO_2 and serve to remove particulate from a gas stream before introduction to a desulfurization scrubber. This latter aspect will be investigated as a part of the FGC Demo project mentioned previously.

Fugitive Emissions Control

Fugitive emissions are major sources of uncontrolled trace metal and SO_2 emissions to the workplace and to the general environment. Several studies underway address the problems of particulate (including volatile and (other) toxic trace elements) and SO_2 fugitives control. This particular area is of crucial interest; because of the high cost potential for controlling fugitives to meet OSHA and EPA standards, the need for control may pose significant constraints upon the use of existing reverb/converter technology. Suitable control techniques or alternative process technologies are needed.

We began a comprehensive evaluation of existing and proposed process control systems and process modifications for copper smelters. This study included an examination of secondary hooding for Pierce-Smith Converters, the Hoboken Converter, whole building evacuation, etc. It drew upon data generated by other Agency components and foreign technology employed in Poland, Belgium, and Japan.

The evaluation of the "second edition" of the Polish version of the Hoboken converter is nearing completion. The newest configuration ("third edition") will be investigated to establish how design changes have improved the overall control of fugitive emissions from the converter operations.

In FY 78, we began a joint project with NIOSH to investigate fugitive emissions in the secondary metals industry. This study is currently focusing on secondary lead smelters. EPA and NIOSH are cost sharing the evaluation of fugitive and process emissions at several exemplary lead plants including several located abroad. The first phase of this study is expected to characterize a system believed to be vastly superior to any currently in use in the United States. If this expectation is realized, a cosponsored demonstration of the system with NIOSH and a selected smelter will be considered. We have under development a project with Trepca (Yugoslavia) which will address the problem of fugitive emissions control at a primary lead/zinc smelter under the Special Foreign Currency Program.

- Water Pollution Problems

The effluent limitations for the Best Available Technology Economically Achievable, to be met in 1984, are currently being developed for primary and some secondary metals industries for the beneficiation, and the smelting and refining subcategories. Many technical problems require resolution before the mandates of the Consent Decree, the 1985 zero pollutant discharge goals, and pending pretreatment guidelines can be met. Process wastewaters generated by this industry contain all 13 of the metallic priority pollutants listed in the June 1976 NRDC/EPA Consent Decree as well as cyanides and various organic reagents used for ore concentration. These create special problems where end-of-pipe treatment must be upgraded to permit recycle and reuse of process water. Improved metals removal will increase the problem of sludge disposal. Current technology provides only for landfilling or ponding, both of which can contribute to groundwater pollution and contamination and which are coming under regulation under the Resource Conservation and Recovery Act (i.e., solid waste act).

The wastewater control program is addressing these problems on several fronts. The immediate goals of the pretreatment regulations significantly affect the secondary metals industry, which is located in urban areas and often discharges to POTW's. For this industry, the treatment technology of choice may be precipitation to control heavy metals. To assist in providing a viable technology, the neutralization and precipitation manual will be completed and published. The Neutralization/Precipitation Manual of Practice will provide Regional personnel involved in permit writing and evaluation and industry with guidance on upgrading existing in-plant treatment processes as well as a design aid for new facilities required to meet the pretreatment and BAT regulations. The design parameters required to meet these less immediate 1984 BAT goals for primary and secondary plants will be available through this manual.

Meeting the Agency's goal of zero discharge of pollutants to surface waters by 1985 poses more difficult problems for the nonferrous industry. Recycling treated process wastewater back into the system, often to the ore concentrating stage, can be utilized on a limited basis. However, when total recycle is used with small amounts of make-up water, quantities of dissolved metals remain in the treated water at levels sufficient to adversely affect ore beneficiation. To remedy this, the use of sulfide precipitation has been investigated, on a limited basis, for treatment of various acidic metal-bearing wastewaters from nonferrous and metal finishing processes (charged membrane ultrafiltration and sulfide precipitation). The pilot scale studies show that a significant improvement in metals removal can be affected by the use of sulfide precipitation. A full-scale system has been installed at Boliden Metall's Ronskaar works in Sweden. An evaluation is underway to identify key design parameters and to determine effluent quality attainable by this treatment mechanism. The results of the Boliden monitoring program will document state-of-the-art performance for sulfide precipitation of heavy metals. Field evaluations will be verified by lab studies conducted at the pilot scale facilities. Sludge and H_2S generation will be studied to evaluate sludge recyclability to roasters/reverbs as well as its stability in landfills.

A study of "Best Management Practices" has been undertaken to identify techniques which control water pollution from nonprocess related sources, i.e., runoff from storage piles in-plant boundaries, tailings ponds, etc. This will incorporate data collected on existing tailings ponds, their location, condition and key operating parameters. Exemplary methods used by industry to reduce surface and groundwater contamination from spills and runoff of plant compounds (fugitive discharges) will be identified and documented to assist in evaluating facets of environmental protection not directly related to processes.

Treatability of the priority pollutants will receive major emphasis but only as far as it promotes the transition into a recycle and reuse program for water and solid waste residuals resulting from wastewater treatment. Process water quality criteria necessary for recycle or reuse of water back into flotation circuits will be developed. These will be incident to the treatability studies being conducted at Boliden, University of Kentucky, and in-house at the Test and Evaluation facility. The objective will be to optimize treatment technologies so that treated process water meets industry intake water purity requirements. Technological deficiencies will be identified in this preliminary recycle/reuse program for concentrated R & D funding. Crossmedia impacts, in so far as they present an obstacle to widescale recycle or reuse of water or conflict with RCRA requirements, will be considered in optimizing any system.

- Solid Waste Problems

The objective of the solid waste research program for the nonferrous metals production industry is first to quantitatively define the potential for environmental pollution resulting from the disposal of process wastes and air and water pollution control residuals. This quantification will be followed by a research, development and demonstration control technology program leading to the establishment of environmentally acceptable practices for smelter waste management for those wastes found to have significantly harmful pollution potential.

Major smelter "solid" waste products are slag, flue dusts, and wastewater pollution control residuals. In current smelter practice, slags are sent to an open slag dump which is a specified area for "storage" of slag materials. Flue dusts are either recycled to the process or discarded to a segregated "storage" area for some unspecified use in the future. Wastewater, containing air pollution control residuals (including acid plant blowdown), is either returned to the tailings pond without treatment or is treated for discharge by sludge producing hydroxide precipitation. All of these wastes contain toxic trace elements in varying amounts. The environmental acceptability of current tailings disposal practices is not defined. However, it has been reported that the State of New Mexico is experiencing contamination of groundwater by trace metals, presumably from nonferrous mining and smelting operations.

Waste disposal and the potential environmental damage that the trace elements could cause, either through leaching and entry into ground or surface waters or through entrainment into the air as fugitive dust, are problems requiring attention. Solid waste management technologies are required which will permit the economic recovery of metal values from the waste coupled with techniques which render the residual wastes environmentally harmless.

The first phase of the program is to quantitatively define the potential for environmental pollution from the disposal of smelter process waste and air and water pollution control residuals. The smelter environmental assessments, as a first step in this problem definition exercise, has identified the points of generation and general characteristics of many of the waste materials produced. This study also identified arsenic-bearing flue dust wastes as the first priority problem due to the arsenic content and the potential for environmental contamination. The second step of this program is directed towards a smelter-by-smelter survey of solid waste management problems and practices. This study will lead to a sampling and analysis program designed to test selected smelter wastes for leachibility and environmental stability.

Arsenic Fixation

The second phase of this program is the research, development, and demonstration of control technology suitable for abating pollution from solid waste management activities. In response to the findings of the smelter environmental assessment study regarding the hazard of arsenic bearing flue dust wastes, we initiated a project to develop safe disposal/recovery techniques (Investigation of New Techniques for Control of Smelter Arsenic Bearing Wastes). This project is completing the second year of its lab phase with efforts concentrated on the development of waste fixation techniques and leach tests of the fixed products. We are directing our efforts towards control of arsenic-bearing flue dust wastes because (1) it is a large source of soluble arsenic (2) the results are needed to support the OAQPS planned regulation of arsenic emissions under Section 112 of the CAA.

Resource Recovery

Negotiations for a program with the Institute of Nonferrous Metals are underway (Poland) to study the feasibility of copper smelter waste utilization under the Special Foreign Currency Program.

Air Pollution Control Residuals

In conjunction with our flue gas cleaning projects, we will be investigating: (1) the characteristics of flue gas cleaning wastes; (2) the possiblities for recovery and recycle of metal and sulfur values from the wastes; (3) techniques for safe ultimate disposal of residuals.

Water Pollution Control Residuals

The environmental aspects of water pollution control residuals will be studied in conjunction with the treatability techniques now under development. The program investigating sulfide precipitation has produced results which indicate that certain metals can be selectively removed from the precipitate (sludge) for recovery. We are also exploring direct recycle of sulfide sludge to permit recovery of metal values.

PROGRAM APPROACH

The goals of these R & D efforts are to develop and evaluate pollution control methods that are broadly applicable to the industry, that present cost effective options for meeting regulations, and that minimize or eliminate intermedia transfer of the pollution problem. Because of the federal manpower limitations, we conduct all of our research activities via extramural means, specificially: contracts, research or demonstration grants, and interagency agreements. Our research funding is presented to us each fiscal year in the form of Congressional appropriations. We in EPA's research office then allocate these funds to projects based upon our awareness of technological needs. Thus, our awareness to the most important research needs is essential, since we are limited in our funding level. We feel that industry input to this program is vital to ensure that useful results - not just paper/reports - are obtained from the expenditure of these funds. Industry participation is needed in four areas: (1) in the activities leading to identification and prioritization of research needs; (2) active participation in the formulation of approaches to satisfy these needs; (3) active participation in reducing selected control technologies to practice under in-plant conditions; (4) review of research findings and reports.

The nonferrous metals research program is available for your use whether you are with EPA or with the industrial sector. We have published reports and are currently addressing problem definition and solution through active projects. We can provide you with more detail on any one of these activities if you desire.

If you feel that you have a solution to a pollution problem and would like to have our assistance in proving out this solution, contact us for discussions of information pertinent to your project. We will be glad to work with you in providing background information which we have available and in cosponsoring work for which we can make funds available.

One last point regarding our working together - you may have a potential solution to a problem that is not relevant to the nonferrous industry, but is relevant to other segments of the industrial sector. If this is the case, I can provide a list of EPA researchers who have responsibility for specific industrial areas.

CHARACTERIZATION OF EMISSION CONTROL PROBLEMS AT A MOLYBDENUM ROASTER

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ABSTRACT

The roasting facility of the Molybdenum Company of America was studied. The plant consists of two multi-level hearth furnaces, a high temperature baghouse and a water quench scrubber. Molybdenum disulfide is oxidized to molybdenum trioxide product and sulfur dioxide.

Removal efficiencies of the baghouse and quench-scrubber for heavy metal compounds are 87% and 89% respectively. The low baghouse efficiency during the testing period was due to broken bags. Sulfate contributions to the grain loadings are 0.4% at the roaster exit (600°F), 17% at the baghouse exit (450°F), and 91% at the quench scrubber exit (gas heated to 250°F according to EPA Method 5). Organic material does not contribute significantly to the quench scrubber exit grain loading.

One roaster (1800 lb/hr MoS_2 feed) produces 14000 scfm stack gas. The main discharges to the atmosphere were:

- 1060 lb/hr Sulfur dioxide
- 28 lb/hr Particulates plus sulfates
- 0.1 lb/hr Molybdenum
- 0.08 lb/hr Selenium
- 0.015 lb/hr Lead
- 0.007 lb/hr Iron
- 0.015 lb/hr Organic material

A bench scale wet electrostatic precipitator completely removed the mist which presently forms the plume. A bench scale lime based SO_2 wet scrubber was also operated and the scrubber products analyzed for impurities. The impurities found were mainly introduced by the lime used and were not constituents of the flue gas.

INTRODUCTION

Molycorp's molybdenum sulfide roasting facility at Washington, Pennsylvania, presently processes 10 million lb/year molybdenum. Particulates and sulfur dioxide escaping the multilevel hearth roasters need to be controlled to bring the plant in compliance with local emissions laws.

Particulates are effectively controlled in a baghouse, quench cooler-packed tower arrangement (quench scrubber). A perspective sketch of this equipment arrangement is shown in Figure 1.

The most effective means for SO_2 control needs to be investigated. The present study was made to supply information about the system performance and to supply data for future changes. These goals were met by:

- Characterizing the roasting process with respect to the flow of elements.
- Determining the effectiveness of the baghouse in removal of particulates.
- Measuring the particulate removal efficiency of the quench scrubber.
- Analyzing the quality of the product of a lime based SO_2 scrubber.
- Determining all important baseline data of the present facility.

The main findings of the study are compiled in tables 1 through 5. The results and their implications are described in the following chapters.

ROASTER OPERATION

During the sampling effort (June 23-78) only one roaster was operating. The feed rate was 1800 lb/hr MoS_2 consisting of a mixture of Questa ore (1650 lb/hr) and baghouse recycle dust (150 lb/hr). The exit gas flow was 8500 scfm (70°F, 29.9" Hg) with an average grain loading of 4.68 grains/scf and a SO_2 concentration of 13000 ppm (v/v). A molybdenum trioxide production rate of 1345 lb/hr was calculated.

The analytical results for the roaster stream can be summarized as follows. Semiquantitative survey analyses by Spark Source Mass Spectrometry (Table 5) for 73 elements indicate that Twin Buttes and Questa concentrates show concentrations in the low ppm range for most heavy metals. Exceptions are lead, zinc, copper, iron and manganese. Twin Buttes ore is high in copper (24,000 ppm versus 1000 ppm in Questa ore), where as Questa ore shows a high lead concentration (4300 ppm versus 930 ppm in Twin Buttes ore).

Volatilization of selenium and mercury is indicated during the roasting process. The mercury and selenium flow rates in the feed stream are 2.3 x 10^{-4} lb/hr and 0.15 lb/hr respectively. The corresponding figures in the MoO_3 product are 0.3 x 10^{-4} lb/hr mercury and 0.003 lb/hr selenium. Volatilization at roaster temperatures gives an explanation for the depletion in the product.

Analytical data and flow determinations were checked by material balance for 15 elements. The closure is good for lead, molybdenum, mercury, bismuth, antimony, cadmium, silver, zinc, copper, manganese, and iron. The rhenium concentration was below the detection limit. Marginal closures were found for arsenic and selenium. The balance for nickel does not close. The reason is not obvious.

BAGHOUSE BALANCE

Sampling to establish a baghouse balance was done simultaneously with the roaster sampling. The following baseline conditions were measured:

Roaster Feed: 1650 lb/hr Questa concentrate
150 lb/hr recycle dust
Flow in: 8500 scfm
Flow out: 13100 scfm
Dust collection rate in baghouse: 272 lb/hr
Grain loading in: 4.68 grains/scf
Grain loading out: 0.41 grains/scf
Collection efficiency 86% (corrected for cooling air dilution)

The chemical analysis of all ingoing and outgoing streams showed that the element distribution of the dust collected in the baghouse is similar to that of the ore. The main crystalline phase as determined by X-ray diffraction is MoS_2. Most of the mercury and a large amount of the selenium are not retained in the baghouse.

Good material balance closure was found for: arsenic, lead, molybdenum, mercury, bismuth, antimony, cadmium, silver, zinc, manganese, and iron. Difficulties were encountered in the balance closure for selenium, copper and nickel.

The removal efficiency of the baghouse based on element flow rates was found to be 87%.

QUENCH SCRUBBER CHARACTERIZATION

The samples for the characterization of the Quench Scrubber were collected on June 21-78. The following base line parameters were measured

Roaster Feed: 1295 lb/hr Questa Ore
159 lb/hr Twin Buttes Ore
346 lb/hr Baghouse Recycle Dust
Inlet gas Flow: 12200 scfm
Outlet gas flow: 14400 scfm
Inlet grain loading: 0.98 grains/scf
Outlet grain loading: 0.23 grains/scf

The water pickup was calculated from the water content at the inlet and outlet gas streams to be 5 gallons/min. Part of the grain loading in both inlet and outlet gas streams was due to sulfuric acid mist condensation. Sulfate formation contributed 0.17 grains/scf at the outlet.

The analytical findings indicate that 90% of the mercury escaping the baghouse is retained in the quench scrubber, and that 50% selenium removal is achieved. The collection efficiency calculated from the elemental flow rates at inlet and outlet is 89%. 20 lbs per hour molybdenum are retained. This high value was probably caused by broken bags lowering the efficiency of the baghouse. Major discharges to the atmosphere are 28 lbs/hr solids containing large amounts of sulfates, and 1060 lbs/hr SO_2. Heavy metal discharge with the flue gas amounts to: 0.1 lb/hr molybdenum, 0.08 lb/hr selenium, 0.015 lb/hr lead, 0.007 lb/hr iron, and 0.015 lb/hr organic material.

LIME SCRUBBER - PRODUCT CHARACTERIZATION

The use of a lime scrubber for SO_2 control will produce about 2500 lbs/hr scrubber sludge from the offgases of one roaster. This sludge has to be disposed of in an environmentally acceptable fashion.

Scrubber products with and without use of a wet electrostatic precipitator (WEP) were generated in a bench scale experiment by scrubbing a slip stream of the flue gas.

The wet electrostatic precipitator removed all of the acid mist. The level of impurities in the product showed a strong correlation with the impurities in the lime. The three major components were iron, manganese, and molybdenum.

	Lime	Product with WEP	Product without WEP
Iron	1500 ppm	1550 ppm	1600 ppm
Manganese	72 ppm	95 ppm	70 ppm
Molybdenum	7 ppm	9 ppm	9 ppm

The concentration of the other elements was in the low ppm or ppb range. The heavy elements in the scrubber liquor investigated were in all cases below the drinking and irrigation water standards. These favorable results are partly due to the short residence time of the liquid in the scrubber.

Verification is needed in long term pilot plant studies.

ORGANIC COMPOUNDS IN PLANT STREAMS

The plume at Molycorp's facility was attributed by some earlier investigators to the discharge of organic material introduced by flotation agents. Extractions of selected solids, liquids and filters were, therefore, investigated by gas chromatography using a flame ionization detector. The results of the semiquantitative analyses show the following:

- Discharge rate with flue gas: 0.015 lb/hr
- Quench water inlet: 0.045 lb/hr
- Quench water blow down: 0.052 lb/hr
- Ore feed: 0.1 lb/hr
- Baghouse recycle dust feed 0.0001 lb/hr

The discharge of 0.015 lb/hr organic material can not be the cause of the plume formation.

SULFATE CONTRIBUTION TO GRAIN LOADING MEASUREMENTS

The high oxygen, high SO_2 concentration of the roaster off-gases favors the formation of SO_3 as the flue gas cools. The SO_3 reacts with solids to form sulfates and with water vapor to form sulfuric acid mist. This increases the grain loading results as the gas temperature decreases. Acid dew points are 309°F at the roaster outlet, and 269°F at the baghouse outlet.

Grain loading determinations showed a particulate concentration at the roaster exit (600°F) of 4.68 grains/scf. Analysis of the filter for sulfate indicated a sulfate contribution of 0.4%. An average grain loading of .70 grains/scf was measured at the baghouse exit (450°F). Sulfate contribution accounted for 17% of the particulate matter. Water droplets and acid mist were present in the quench-scrubber exit gases. Water had to be evaporated in order to determine a grain loading. The EPA method 5 was used. Contributions of sulfuric acid mist accounted for 50%, 74% and 91% at sampling temperatures of 380°F, 300°F and 250°F, respectively. These results point to sulfuric acid mist as the cause of the plume formation. The mist was completely removed in a bench scale wet electrostatic precipitator.

BAGHOUSE EFFICIENCY AND PARTICLE SIZING STUDIES

Baghouse efficiency studies showed the baghouse to be operating below expected levels. Efficiencies were found to be 85-90%. Particle sizing showed a very high percentage of large (>2μm) material penetrating the baghouse. This is indicative of unreplaced broken bags. Elements in the vapor phase below 250°F were negligible.

CONCLUSIONS

1. Sulfate particulates and sulfuric acid mist, not organics, contribute most to the grain loading observed in the stack gas.

2. The installation of a wet electrostatic precipitator will eliminate the present plume problems.

3. The disposal of lime/limestone based scrubber byproducts needs to be addressed. Areas of interest are:

    - dumping sites
    - sludge fixation
    - leaching properties

4. The reason for the low baghouse efficiency during the testing period appears to be unreplaced broken bags.

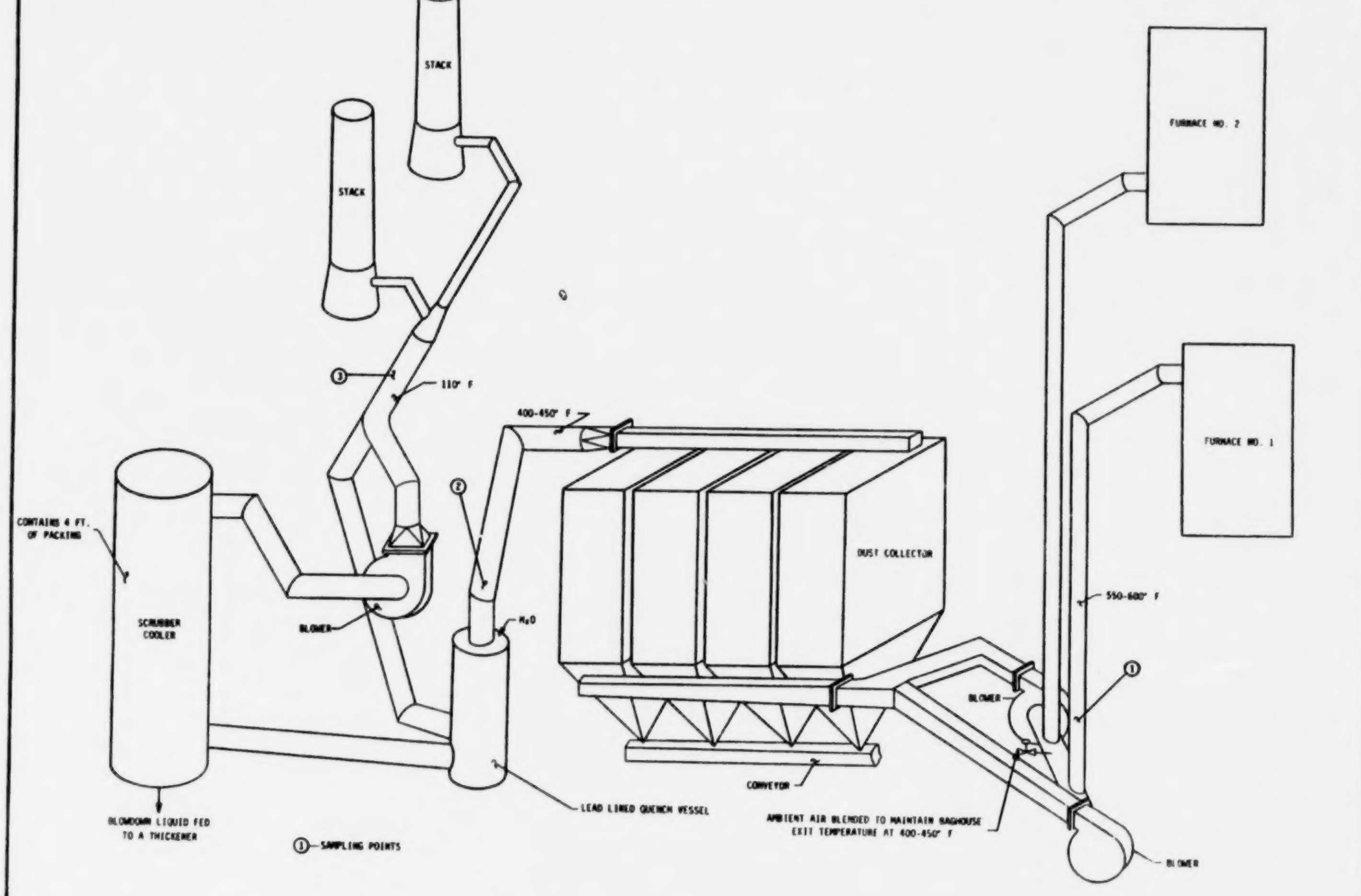
STACK
STACK
FURNACE NO. 2
FURNACE NO. 1
③
110° F
400-450° F
②
H₂O
DUST COLLECTOR
550-600° F
①
BLOWER
CONTAINS 4 FT. OF PACKING
SCRUBBER COOLER
BLOWER
BLOWDOWN LIQUID FED TO A THICKENER
LEAD LINED QUENCH VESSEL
CONVEYOR
AMBIENT AIR BLENDED TO MAINTAIN BAGHOUSE EXIT TEMPERATURE AT 400-450° F
BLOWER
① — SAMPLING POINTS

FIGURE 1:
DUST COLLECTION SYSTEM AT MOLYCORP

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TABLE 1. SUMMARY OF SAMPLING EFFORT JUNE-1978

Date	Site[1]	Sample	Procedure	Comments
Roaster Balance				
6/23	1	2 Grain Loading	EPA-5	
		1 $SO_2 - SO_3$	EPA-8	
		1 Vapor Phase Trace Elements	Acid/Base Train	
		2 Total Elements	Radian WEP	
	4	Questa Ore Bin 6	24 Hr Composite	
		Questa Ore Bin	24 Hr Composite	
	5	Baghouse Recycle Dust	24 Hr Composite	
	8	Product Molybdenum Oxide	24 Hr Composite	
Baghouse Balance				
6/22	2	2 Grain Loadings	EPA-5	
		1 Vapor Phase Trace Elements	EPA-8	
		1 Particle Size	In-stack Andersen	
	5	1 Baghouse Recycle Dust	24 Hr Composite	
6/23	1	2 Grain Loadings	EPA-5	
		1 $SO_2 - SO_3$	EPA-8	
		1 Vapor Phase Trace Element	Acid/Base Train	
		2 Total Elements	Radian WEP	
	2	1 Grain Loading	EPA-5	
		1 $SO_2 - SO_3$	EPA-8	
		1 Total Elements	Radian WEP	
	5	1 Baghouse Dust	24 Hr. Composite	
6/24	1	1 Particle Size	In-stack Brinks	
Quench Scrubber Efficiency				
6/20	3	2 Grain Loadings	EPA-5	Filters at 380°F
		2 $SO_2 - SO_3$	EPA-8	
		2 Particle Size	Out-of-Stack Andersen	Sampler at 300°F
	6	1 Quench Scrubber	Injection Water	Grab Sample & Flow Rate
	7	1 Quench Scrubber	Blow Down Water	Grab Sample & Flow Rate
6/21	2	2 Grain Loading	EPA-5	In-Stack Filter
		2 $SO_2 - SO_3$	EPA-8	
		1 Total Elements	Radian WEP	
	3	1 Grain Loading	EPA-5	Filter at 300°F
		1 $SO_2 - SO_3$	EPA-8	
		2 Total Element	Radian WEP	
6/22	2	1 Grain Loading	EPA-5	In-Stack Filter
		1 Vapor Phase Trace Elements	Acid/Base Train	
		1 Total Elements	Radian WEP	
		1 Particle Size	In-Stack Andersen	
	3	1 Grain Loading	EPA-5	Filter At 250°F
		1 Vapor Phase Trace Elements	Acid/Base Train	
	6	1 Quench Injection Water	Grab Sample & Flow Rate	
	7	1 Quench Blow Down Water	Grab Sample & Flow Rate	
Scrubber Product Characterization				
6/20	3	1 Radian Scrubber Product		Without Radian WEP
6/23	3	1 Radian Scrubber Product		With Radian WEP
Baghouse Efficiency Studies				
6/22	2	1 Grain Loading	EPA-5	
		1 Particle Size	In-Stack Andersen	
6/23	1	2 Grain Loadings	EPA-5	
		1 $SO_2 - SO_3$	EPA-8	
	2	1 Grain Loading	EPA-5	
		1 $SO_2 - SO_3$	EPA-8	
6/24	1	1 Particle Size	In-Stack Brinks	
6/26	1	2 Grain Loadings	EPA-5	
		2 $SO_2 - SO_3$	EPA-8	
		1 Particle Size	In-Stack Brinks	
	2	2 Grain Loadings	EPA-5	
		2 $SO_2 - SO_3$	EPA-8	
		3 Particle Size	In-Stack Andersen	
6/27	1	1 Grain Loading	EPA-5	
		1 $SO_2 - SO_3$	EPA-8	
		1 Particle Size	In-Stack Brinks	
	2	1 Grain Loading	EPA-5	
		1 $SO_2 - SO_3$	EPA-8	
		Particle Size	In-Stack Andersen	
Atomspheric Emissions				
6/20	3	1 Grain Loading SO_4^{-2} as H_2SO_4 Organics	EPA-5	Dual Filters at 380°F Vertical Traverse
		1 Grain Loading SO_4^{-2} as H_2SO_4 Organics	EPA-5	Dual Filters at 365°F Horizontal Traverse
		1 $SO_2 - SO_3$	EPA-8	380°F
		1 $SO_2 - SO_3$	EPA-8	365°F
6/21	2	2 Grain Loading SO_4^{-2} as H_2SO_4	EPA-5	Dual In-Stack Filters Plus Out of Stack Filter at 300°F
		2 $SO_2 - SO_3$	EPA-8	
	3	1 Grain Loading SO_4^{-2} as H_2SO_4 Organics	EPA-5	Dual Filters at 300°F
		1 $SO_2 - SO_3$	EPA-8	
6/22	3	1 Grain Loading SO_4^{-2} as H_2SO_4 Organics	EPA-5	Dual Filters at 250°F

[1] Sample Site
1 Between Roaster and Baghouse
2 Between Baghouse and Quench Scrubber
3 After Quench Scrubber
4 Concentrate Feed at Feeder
5 Baghouse Dust Feed at Feeder
6 Recycle Pump By-Pass
7 Quench Scrubber Blow Down Drain
8 Product Stream From Roaster

TABLE 2. SUMMARY OF BASE LINE MEASUREMENT RESULTS

EXPERIMENTAL RESULTS		June-20	June-21	June-22	June-23	June-26	June-27
1. Ore feed rates							
• Questa 6 lbs/hr		ND	1295	1470	1650	ND	ND
• Twin Buttes lbs/hr		ND	159	180	0	ND	ND
• Baghouse Dust Recycle		ND	346	150	150	ND	ND
TOTAL FEED RATE		ND	1800	1800	1800		
2. MoO_3 Production rate (lb/hr)		ND	1430	1450	1380	ND	ND
3. Gas Flow							
• Roaster Exit scfm		ND	ND	ND	8400	8150	9200
• Baghouse Exit scfm		ND	11950	11500	13100	12650	13000
• Stack Discharge scfm		13600	14400	14700	ND	ND	ND
4. Grain Loading							
• Roaster Exit grains/scf		ND	ND	ND	4.68	5.50	7.33
• Baghouse Exit grains/scf		ND	.98	.49	.41	.98	.71
• Stack grains/scf		.26	.23	.43	ND	ND	ND
5. Gas Composition							
• Roaster Outlet - SO_2 ppm		ND	ND	ND	13500	13550	12700
- SO_3 ppm		ND	ND	ND	90	87	114
H_2O %		ND	ND	ND	3.5	4.8	4.4
• Baghouse Outlet - SO_2 ppm		ND	6100	ND	4560	7360	5980
SO_3 ppm		ND	20	ND	7	20	15
H_2O %		ND	3.3	3.2	2.8	4.3	3.5
• Stack Discharge - SO_2 ppm		6500	7360	ND	ND	ND	ND
SO_3 ppm		46	18	ND	ND	ND	ND
H_2O %		8.8	8.9	8.9	ND	ND	ND
6. Baghouse Dust Catch lb/hr		ND	219	202	275	ND	ND
7. Intake Quench Water gal/min		ND	32	35	ND	ND	ND
8. Quench Water Blowdown gal/min		ND	26.6	30	31.9	ND	ND
9. Particle size distribution (% less than indicated size)[1]					(June-24)		
• Brinks Roaster Exit							
- Cyclone catch	<2.1 um				24	21	26
- Plate 1	<1.2				17	16	20
- Plate 2	<0.50				12	12	15
- Plate 3	<0.35				8.5	9.4	11
- Plate 4	<0.22				6.7	7.0	8.8
- Plate 5	<0.14				4.4	4.8	6.6
• Anderson Baghouse Exit							
- Plate 1	<7.4 um				(<5.8 um-80)	88 72 70	69
- Plate 2	<6.8				(<5.4 -60)	76 65 58	53
- Plate 3	<4.2				(<3.3 -45)	64 48 51	40
- Plate 4	<2.8				(<2.7 -36)	50 43 42	34
- Plate 5	<1.5				(<1.1 -25	37 30 33	29
- Plate 6	<0.78				(<0.59 -15)	25 20 24	23
- Plate 7	<0.37				(<0.27 -6.8)	12 10 17	19
- Plate 8	<0.18				(<0.12 -1.0)	1.1 3.2 11	17
- Filter							
• Anderson Stack							
- Plate 1	<8.0 um	92 82					
- Plate 2	<7.4	87 65					
- Plate 3	<4.6	83 50					
- Plate 4	<3.1	81 38					
- Plate 5	<1.6	79 24					
- Plate 6	<0.88	77 13					
- Plate 7	<0.44	76 4					
- Plate 8	<0.23	71 0.2					
- Filter							

TABLE 3. SUMMARY OF QUANTITATIVE ANALYTICAL RESULTS

Sample Description	ROASTER BALANCE				BAGHOUSE BALANCE		HUMIDIFIER SCRUBBER EFFICIENCY				LIME & SCRUBBER PRODUCT IMPURITIES		
Element	Questa Ore ppm	Twin Buttes Ore ppm	MoO_3 Product ppm	WEP Site 1 June-23-78 µg/acf*	Baghouse Dust ppm	WEP Site 2 June-23-78 µg/acf*	WEP Site 2 June-21-78 µg/acf*	WEP Site 3 June-21-78 µg/acf*	Inlet Water µg/ml	Outlet Water µg/ml	Lime ppm	Scrubber Product Without Prescrubber ppm	Scrubber Product With Prescrubber ppm
Arsenic	12	9.9	19	15	48	1.67	1.60	.38	1.3	1.62	1	1	1
Lead	4300	980	4800	1780	6000	139	120	7.8	10.2	10.1	<1	1	1
Molybdenum	470000	360000	540000	149000	490000	11500	11400	55	3040	3810	7	9	9
Mercury	15	.15	.023	.22	23	.11	.11	.016	.032	.061	<.005	<.005	<.005
Selenium	56	122	2.0	157	400	89	90	44	6.95	10.2	.3	1	.5
Bismuth	280	63	160	113	350	4.8	4.5	.22	1.51	3.67	<.01	<.01	<.01
Antimony	21	ND	19	7.0	27	.59	.55	.03	.13	0.39	.1	1	1
Cadmium	18	14	20	7.6	22	.70	.70	.16	.55	0.67	<.01	<.01	<.01
Silver	38	62	49	14.2	52	1.23	1.20	.39	<.05	<.05	.05	.07	.09
Zinc	1500	240	1700	485	1900	45.3	45	.26	43.1	51.5	5	25	19
Nickel	96	20	59	8.4	55	.78	75	.078	1.89	2.22	4	15	5
Copper	950	24000	1500	401	3700	19.3	19	.78	158	194	1.5	3	2.2
Manganese	730	840	890	197	770	14.0	13	.70	39	46	72	70	95
Iron	11800	32500	17000	3750	17800	31.4	31	3.9	850	1000	1500	1600	1550
Rhenium	<10	<10	<10	<1	<10	<1	<1	<1	<1	<1	<10	<10	<10

*µg/acf of gas sampled

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TABLE 4. SUMMARY OF ELEMENTAL FLOW RATES IN ROASTER STREAMS

Sample Element	ROASTER STREAMS 6-23 Queste Feed (1650 lb/hr) lb/hr	Dust Feed (150 lb/hr) lb/hr	MoO_3 Product (1345 lb/hr) lb/hr	Roaster Dust (341 lb/hr) lb/hr	BAGHOUSE STREAMS 6-23 Baghouse Dust (272 lb/hr) lb/hr	Baghouse Gas & Dust (46 lb/hr) lb/hr	HUMIDIFIER SCRUBBER STREAMS JUNE-21-78 Baghouse Gas & Dust (103 lb/hr) lb/hr	Quench Off-Gases (16 lb/hr) lb/hr	Quench Water Inlet (31 gal/min) μg/g	Quench Water Outlet 26 gal/min μg/g	LIME & SCRUBBER PRODUCT Lime (1110 lb/hr) lb/hr	Gypsum Without Prescrubber (2590 lb/hr) lb/hr	Gypsum With Prescrubber (2590 lb/hr) lb/hr
Arsenic	020	.007	.025	.017	.013	.003	.003	.0007	.020	.022	1 x 10^{-3}	3 x 10^{-6}	3 x 10^{-4}
Lead	7.1	.90	6.44	2.00	1.64	.24	.21	.015	.17	.134	<1 x 10^{-3}	<3 x 10^{-3}	3 x 10^{-6}
Molybdenum	780	74	725	158	134	20	19.8	.101	48	51	8 x 10^{-3}	23 x 10^{-3}	23 x 10^{-3}
Mercury	2 x 10^{-6}	.3 x 10^{-6}	.3 x 10^{-5}	2.5 x 10^{-6}	.6 x 10^{-6}	2.0 x 10^{-6}	2.0 x 10^{-6}	.3 x 10^{-6}	5 x 10^{-6}	8 x 10^{-6}	<6 x 10^{-6}	<13 x 10^{-6}	13 x 10^{-6}
Selenium	.092	.060	.003	.18	.11	.16	.16	.08	.11	14	3 x 10^{-5}	30 x 10^{-6}	15 x 10^{-6}
Bismuth	.46	.05	.21	.13	.10	.008	.008	4 x 10^{-6}	.02	.05	<1 x 10^{-6}	<3 x 10^{-6}	3 x 10^{-5}
Antimony	.035	.004	.025	.038	.007	.001	.001	.6 x 10^{-6}	.005	.005	1 x 10^{-5}	3 x 10^{-6}	3 x 10^{-6}
Cadmium	030	.003	.027	.039	.006	.001	.001	3 x 10^{-6}	.008	.009	<1 x 10^{-6}	<3 x 10^{-5}	3 x 10^{-5}
Silver	.063	.008	.066	.016	.014	.002	.002	7 x 10^{-6}	<.0008	<.0007	6 x 10^{-6}	18 x 10^{-5}	23 x 10^{-6}
Zinc	2.48	.29	2.28	.55	.52	.08	.08	5 x 10^{-6}	.68	.685	6 x 10^{-3}	65 x 10^{-3}	49 x 10^{-3}
Nickel	.16	.008	.079	.039	.015	.001	.001	<1 x 10^{-6}	.03	.030	4 x 10^{-3}	39 x 10^{-3}	13 x 10^{-3}
Copper	1.6	.56	2.0	.65	1.0	.03	.033	1.5 x 10^{-3}	2.5	2.6	2 x 10^{-3}	8 x 10^{-3}	6 x 10^{-3}
Manganese	1.2	.12	1.19	.22	.21	.02	.023	1.3 x 10^{-3}	.62	.612	8 x 10^{-3}	18 x 10^{-2}	25 x 10^{-3}
Iron	20	2.7	22.8	4.22	4.88	.05	.054	7.3 x 10^{-3}	13.2	13.3	1.7	4.1	4.0
Rhenium	<.02	<.002	<.01	<.3	<.003	<.005	<.005	<.0003	<1.3	<.04	01	<.03	<.03

TABLE 5. SUMMARY OF SURVEY ANALYSES BY SPARK SOURCE MASS SPECTROMETRY

Element	Twin Buttes Ore ppm	Questa Ore ppm	MoO$_3$ Product ppm	June 23 WEP Site 1 μg/acf	Baghouse Dust ppm	June 23 WEP Site 2 μg/acf*	June 21 WEP Site 2 μg/acf*	June 21 WEP Site 3 μg/acf*	Quench Water In μg/ml	Quench Water Out μg/ml	Lime ppm	Scrubber Product Without WEP Prescrubber ppm	Scrubber Product With WEP Prescrubber ppm
Uranium	8	15	20	7	10	-	.3	-	.1	1	7	<2	<2
Thorium	3	22	32	7	*17	-	.3	-	.2	3	-	-	-
Bismuth	90	40	77	29	300	.3	3	.04	.4	.1	-	-	-
Lead	430	>1000	>1000	>360	>1000	3	26	.2	4	.3	1	1	1
5 Thallium	1	.7	11	3.5	.7	-	.02	-	.01	.02	-	-	-
Mercury	NR	NR	-	NR	NR	NR	NR	NR	NR	NR	NR	NR	NR
Gold	.2	-	<.2	-	<.1	-	-	-	-	<.04	-	-	-
Platinum	-	-	< 2	-	-	-	-	-	-	-	-	-	-
Iridium	-	-	-	-	-	-	-	-	-	-	-	-	-
10 Osmium		-	-	-	-	-	-	-	-	-	-	-	-
Rhenium	140	11	2	<.2	13	5	4	*3	.2	2	-	-	-
Tungsten	26	5	10	3.5	9	-	.4	-	.6	3	-	-	-
Tantalum	< 4	<2	-	-	<4	<.4	<.07	<.8	-	<.08	-	-	-
Hafnium	-	-	-	-	-	-	-	-	-	-	-	-	-
15 Lutetium	-	-	-	-	-	-	-	-	-	.02	-	-	-
Ytterbium	-	-	1	-	-	-	-	-	-	.1	-	-	-
Thulium	-	-	.2	-	-	-	-	-	-	.01	-	-	-
Erbium	.5	.9	.7	-	.5	-	-	-	.005	.1	-	-	-
Holmium	.6	1	1	-	.6	-	-	-	.008	.1	-	-	-
20 Dysprosium	*1	2	2	-	1	-	-	-	.02	.2	-	-	-
Terbium	.2	.4	.7	-	<.2	-	-	-	.004	.05	-	-	-
Gadolinium	.9	1	3	-	1	-	-	-	.02	.1	-	-	-
Europium	.5	.5	.9	.1	.5	-	-	-	.008	.1	-	-	-
Samarium	2	2	9	.7	2	-	.06	-	.1	.9	-	-	-
25 Neodymium	6	6	25	3.7	13	-	.2	-	2	5	1	-	.8
Praseodymium	*2	1	5	1.8	6	-	.1	-	1	2	.3	.3	.3
Cerium	14	12	32	21	30	.1	.4	.05	3	>10	3	2	2
Lanthanum	12	43	58	21	30	.06	.4	.07	8	>10	2	2	1
Barium	31	44	60	18	56	1	.4	.2	.02	.2	10	6	6
30 Cesium	3	1	1	.3	2	-	.03	-	<.001	.02	-	-	-
Iodine	35	-	10	3.5	<5	-	.2	-	.01	.05	-	.7	-
Tellurium	22	38	42	25	220	.7	.5	<.1	< 1	1	-	-	-
Antimony	41	72	78	14	91	.4	1	.04	.2	.2	.7	-	-
Tin	26	19	51	1.8	47	.6	.6	.1	.1	.2	-	-	-
35 Indium	STD	STD	STD	STD	STD	STD	STD	STD	STD	STD	STD	STD	STD

Continued

*μg/acf of gas sample
NR - Not reported
STD - Standard

TABLE 5. (CONTINUED) SUMMARY OF SURVEY ANALYSES
BY
SPARK SOURCE MASS SPECTROMETRY

Element	Twin Buttes Ore ppm	Questa Ore ppm	MoO₃ Product ppm	June 23 WEP Site 1 µg/acf	Baghouse Dust ppm	June 23 WEP Site 2 µg/acf*	June 21 WEP Site 2 µg/acf*	June 21 WEP Site 3 µg/acf*	Quench Water In µg/ml	Quench Water Out µg/ml	Lime ppm	Scrubber Product Without WEP Prescrubber ppm	Scrubber Product With WEP Prescrubber ppm
Cadmium	170	67	330	25	85	.04	1	-	.7	8	-	-	-
Silver	120	220	75	32	160	>.2	>.2	>.2	.08	.5	.7	.9	.6
Palladium	-	-	-	-	-	-	-	-	-	-	-	-	-
Rhodium	-	-	-	-	-	-	-	-	-	-	-	-	-
Ruthenium	-	-	-	-	-	-	-	-	-	-	-	-	-
Molybdenum	>1000	>1000	>1000	>360	>1000	>95	>90	>90	>10	>10	10	5	2
Niobium	3	11	23	3.5	15	.04	.5	-	.3	1	.8	.5	<.4
Zirconium	19	27	29	1.4	19	.6	.2	.60	.1	1	4	3	3
Yttrium	17	14	16	2.5	17	.07	.2	-	.2	1	2	2	5
Strontium	47	33	26	2.1	83	.2	.6	.08	3	7	430	150	150
Rubidium	25	8	13	3.2	10	>.01	.2	>.01	.2	.3	.4	<.1	<.1
Bromine	10	.8	4	.7	3	-	.04	.04	.02	.1	1	<.6	<1
Selenium	290	110	7	2.1	480	5	9	9	2	>10	1	9	9
Arsenic	49	43	47	7.1	81	1	2	.9	1	.5	3	2	3
Germanium	7	1	1	.1	7	.05	.2	>.04	.02	.1	-	5	3
Gallium	10	5	15	2.5	4	.02	.1	.03	.1	.4	1	.3	.5
Zinc	>1000	>1000	>1000	>360	>1000	3	27	*2	>10	>10	9	35	35
Copper	>1000	930	>1000	>360	>1000	6	90	.7	>10	>10	3	6	7
Nickel	210	210	75	14	48	>95	2	1	2	7	8	29	9
Cobalt	22	40	36	.7	22	.7	1	.04	2	8	.4	3	3
Iron	>1000	>1000	>1000	>360	>1000	50	>90	27	>10	>10	>1000	>1000	>1000
Manganese	>1000	>1000	>1000	290	>1000	2	8	.3	>10	>10	100	22	36
Chromium	6	24	52	3.5	6	1	.8	.3	1	>10	9	1	2
Vanadium	22	39	42	3.5	22	2	.5	.04	.5	1	12	5	7
Titanium	INT	INT	INT	INT	INT	INT	INT	INT	INT	INT	5	62	50
Scandium	5	12	3	.1	2	>.02	.1	>.03	.08	.7	1	2	<.2
Calcium	>1000	>1000	>1000	>360	>1000	>95	>90	>90	>10	>10	>1000	>1000	>1000
Pottassium	480	>1000	>1000	>360	>1000	10	62	27	>10	>10	120	87	87
Chlorine	13	8	42	-1	13	.09	.5	.1	.3	>10	28	53	80
Sulfur	>1000	>1000	>1000	>360	>1000	>95	>90	>90	>10	>10	62	>1000	>1000
Phosphorus	50	>1000	>1000	>360	670	7	27	36	>10	>10	86	91	91
Silicon	>1000	>1000	>1000	290	>1000	30	9	18	>10	>10	>1000	>1000	>1000
Aluminum	>120	200	>1000	70	>120	>20	>18	4	>2	>5	>200	>210	> 210
Magnesium	>120	>1000	>1000	>360	>1000	40	>90	9	>10	>10	>1000	>1000	>1000
Sodium	>300	20	>1000	>210	50	>50	44	18	>5	>10	26	18	27
Fluorine	>1000	~320	~160	~140	~140	~.3	3	~.2	~2	~3	560	~190	~250
Oxygen	NR	NR	NR	NR	NR	NR	NR	NR	NR	NR	NR	NR	NR
Nitrogen	NR	NR	NR	NR	NR	NR	NR	NR	NR	NR	NR	NR	NR
Carbon	NR	NR	NR	NR	NR	NR	NR	NR	NR	NR	NR	NR	NR
Boron	.2	.3	.5	.7	.7	.4	.1	>.01	2	6	.8	.5	4
Beryllium	-	-	-	1.4	-	-	-	-	-	-	-	-	-
Lithium	4	7	11	3.5	4	.07	.04	.01	.7	.8	1	.6	.7
Hydrogen	NR	NR	NR	NR	NR	NR	NR	NR	NR	NR	NR	NR	NR

*µg/acf of gas sampled
NR - Not Reported
STD - Standard

PRESENT AND FUTURE CONTROL OF FUGITIVE EMISSIONS IN THE PRIMARY NONFERROUS METALS INDUSTRY

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ABSTRACT

The functions of the process equipment used in the primary copper industry; sources and characteristics of fugitive emissions; methods currently employed for control of fugitive emissions and their collective efficiencies; future considerations for controlling fugitives with existing technology and new concepts in process and/or control equipment; and costs and energy requirements for existing and future techniques for fugitive emission controls are reviewed.

INTRODUCTION

This paper deals with fugitive emissions from the primary copper smelter as it exists today and takes a look into the future with respect to known technology and a combination of ideas from other industries to minimize these fugitive emissions. The thoughts and ideas presented are those of the PEDCo author. Their presentation here does not represent an approval of the U.S. Environmental Protection Agency.

Figure 1, which represents a conventional smelter layout, shows the feed being conveyed to a multihearth roaster, then transported to a reverberatory furnace. The matte from the reverberatory furnace is charged to a Peirce-Smith converter, and the slag is sent to a disposal area. The blister copper from the converter is ladled to an anode furnace, and the converter slag is returned to the reverberatory furnace.

The fugitive emission figures shown on the following tables were estimated from information supplied in reports and papers presented at other meetings. These values can differ substantially from plant to plant. Before making these estimates, we attempted to secure published values that had been arrived at through field measurements, but several inquiries proved unsuccessful. The following are the reports and papers from which the information was extracted:

The Bureau of Mines report RI 7957, entitled "Cost of Producing Copper from Chalcopyrite Concentrate as Related to SO_2 Emission Abatement," by Masami Hayashi, H. Dolezal, and J.H. Bilbrey, Jr. (1974).

The Radian Corporation report, entitled "Pollution Control and Heat Recovery from Non-Ferrous Smelters," Volume II.

An Abstract, entitled "Oxygen Smelting in the Noranda Process," by J.B.W. Bailey, R.R. Beck, G.D. Hallett, C. Washburn, And A.J. Weddick. Presented at the AIME annual meeting in New York City in February 1975.

An abstract, entitled "New Developments in Outokumpu Flash Smelting Methods," by S.U. Harkki and J.T. Juusela. Presented at the AIME annual meeting in Dallas in February 1974.

The Bureau of Mines report was the primary source of the emission values, but we modified the numbers based on producing equivalent tonnages of blister copper.

From a review of currently used methods for controlling fugitive emissions, it appears such methods consist primarily of hooding and building enclosures. The efficiency of these systems, when retrofitted to existing reverberatory furnaces or converters, is limited in all phases of the process operation. For example, a Peirce-Smith converter equipped with fixed or secondary fugitive hooding will capture at the most a small percentage of the fugitive emissions created during the charging of matte or slagging or blister copper pouring as shown in Figures 15, 16, and 17 (see Appendix B-1). When slag from the Pierce-Smith converter is being charged, the reverberatory furnace permits the fugitive emissions to be discharged to the surrounding area and to the roof monitors.

Table 1[a]

	Estimated Fugitive Emissions			
	SO_2	Cu	Fe	Others
Multihearth (Figure 2)	-	-	-	-
Reverberatory (Figure 3)	4.2	0.16	0.10	0.10
Peirce-Smith Converter (Figure 4)	6.5	0.16	0.21	0.11

Table 2[a]

Fluid-bed Roaster (Figure 5)	-	-	-	-
Reverberatory (Figure 6)	4.2	0.16	0.09	0.12
Peirce-Smith Converter (Figure 7)	6.5	0.16	0.21	0.13

Table 3[a]

Noranda (Figure 8)	2.9	0.11	0.07	0.04
Peirce-Smith Converter (Figure 9)	1.5	0.03	0.33	0.12

Table 4[a]

Dryer (Figure 10)	-	-	-	-
Outokumpu (Figure 11)	2.9	0.15	0.15	0.07
Peirce-Smith Converter (Figure 12)	2.1	0.03	0.38	0.08

Table 5[a]

Dryer (Figure 10)	-	-	-	-
Electric Furnace (Figure 13)	3.0	0.15	0.07	0.04
Peirce-Smith Converter (Figure 14)	6.5	0.19	0.21	0.06

[a] These tables provide a comparison of the different process systems. Percentage shown could vary substantially from plant to plant.

Percentage of sulfur charged to the process equipment, expressed as SO_2.

Percentage of the total charged to the process equipment.

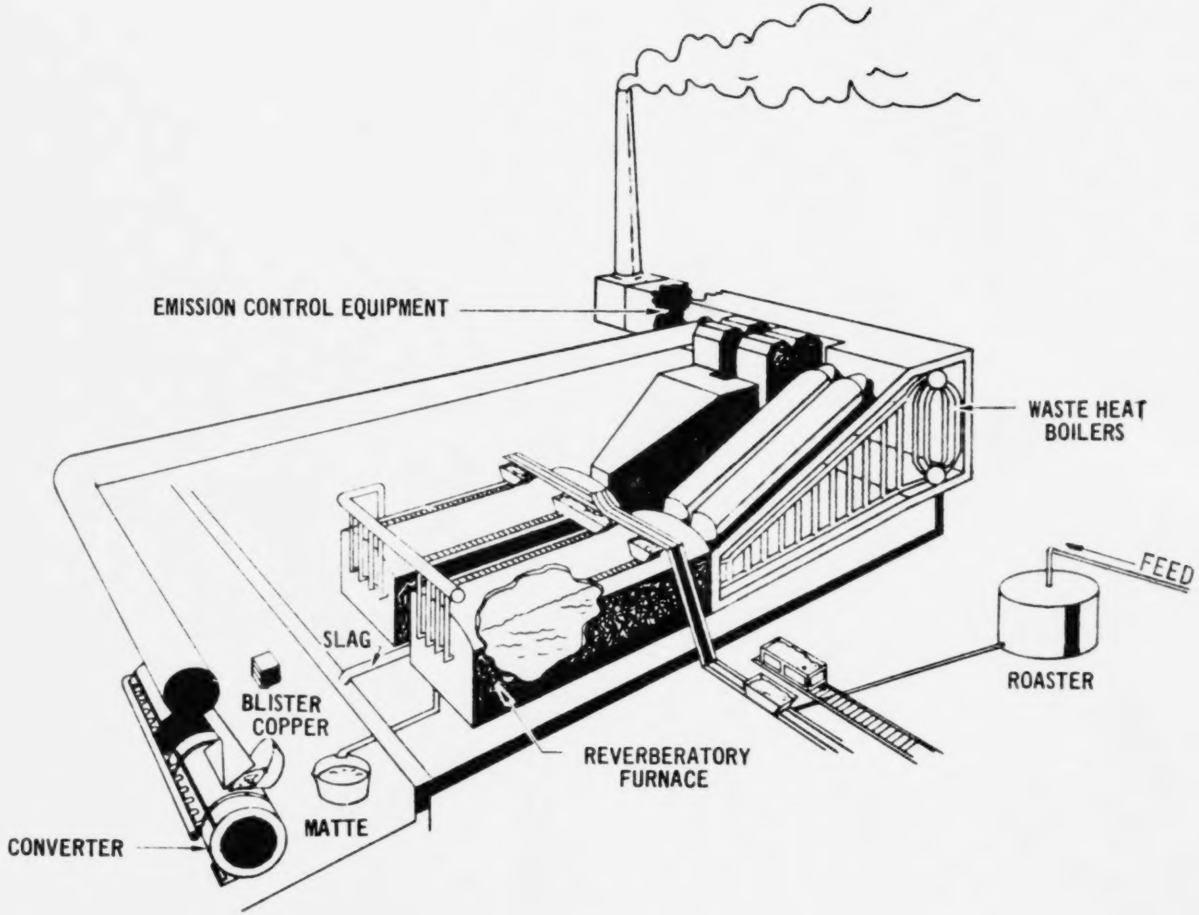
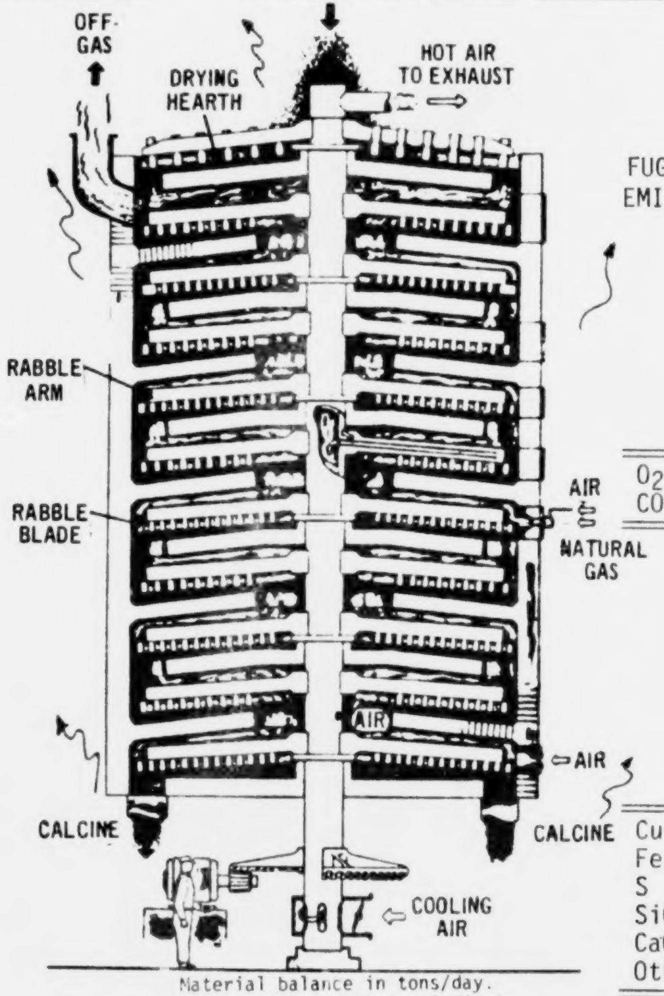

EMISSION CONTROL EQUIPMENT
WASTE HEAT BOILERS
FEED
ROASTER
SLAG
BLISTER COPPER
REVERBERATORY FURNACE
MATTE
CONVERTER

Figure 1. Conventional smelter layout.*
Source: "Background Information for New Source Performance Standards: Primary Copper, Zinc, Lead Smelters" Volume I: Proposed Standards. EPA 450/2-74-002a. October 1974.

SO_2	383.0	(54.8)	7.0% vol.
CO_2	53.5	(7.6)	

Feed		
Cu	308.6	(44.1)
Fe	308.6	(44.1)
S	382.9	(54.7)
SiO_2	102.9	(14.7)
CaO	5.7	(0.8)
Other	34.2	(4.9)

Flux		
CaO	49.0	(7.0)
SiO_2	97.1	(13.9)
CO_2	37.3	(5.3)
Other	12.2	(1.7)


OFF-GAS
DRYING HEARTH
HOT AIR TO EXHAUST
FUGITIVE a EMISSIONS
RABBLE ARM
RABBLE BLADE
AIR
NATURAL GAS
AIR
AIR
CALCINE
CALCINE
COOLING AIR

O_2	191.5	(27.4)
CO_2	16.2	(2.3)

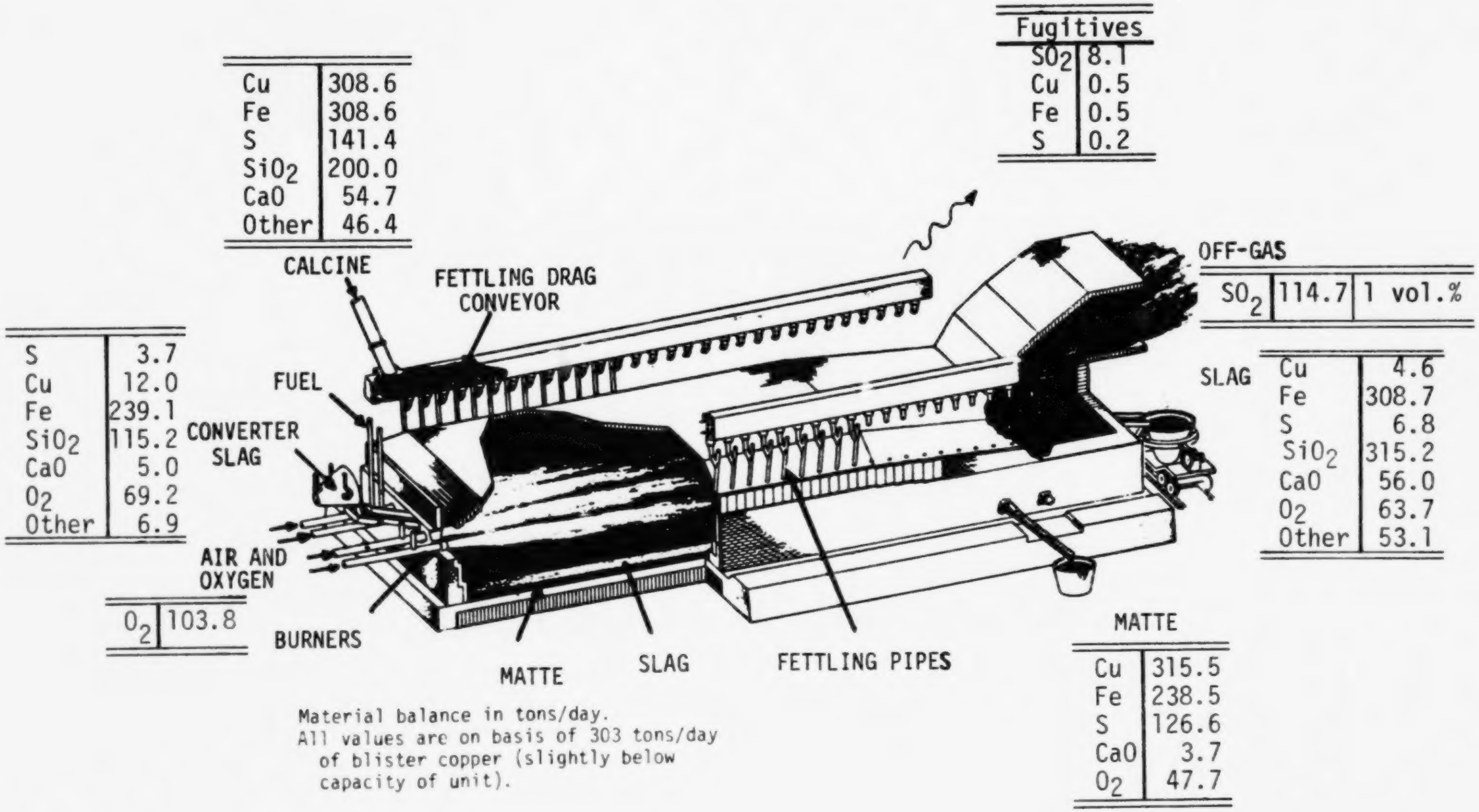
Cu	308.6	(44.1)
Fe	308.6	(44.1)
S	191.4	(27.3)
SiO_2	200.0	(28.6)
CaO	54.7	(7.8)
Other	46.4	(6.6)

Material balance in tons/day.
Numbers in parentheses indicate capacity of each unit.
Other values are based on 383 tons/day of blister copper.

[a] Values not available.

Figure 2. Multiple-hearth roasting furnace.*

*Source: Background Information; EPA 450/2-74-002a, October 1974.


CALCINE
FETTLING DRAG CONVEYOR
FUEL
CONVERTER SLAG
AIR AND OXYGEN
BURNERS
MATTE
SLAG
FETTLING PIPES
Fugitives
OFF-GAS
SLAG
MATTE

Calcine:

Cu	308.6
Fe	308.6
S	141.4
SiO_2	200.0
CaO	54.7
Other	46.4

Converter slag:

S	3.7
Cu	12.0
Fe	239.1
SiO_2	115.2
CaO	5.0
O_2	69.2
Other	6.9

Air and oxygen:

O_2	103.8

Fugitives	
SO_2	8.1
Cu	0.5
Fe	0.5
S	0.2

OFF-GAS

SO_2	114.7	1 vol.%

SLAG

Cu	4.6
Fe	308.7
S	6.8
SiO_2	315.2
CaO	56.0
O_2	63.7
Other	53.1

MATTE

Cu	315.5
Fe	238.5
S	126.6
CaO	3.7
O_2	47.7

Material balance in tons/day.
All values are on basis of 303 tons/day of blister copper (slightly below capacity of unit).

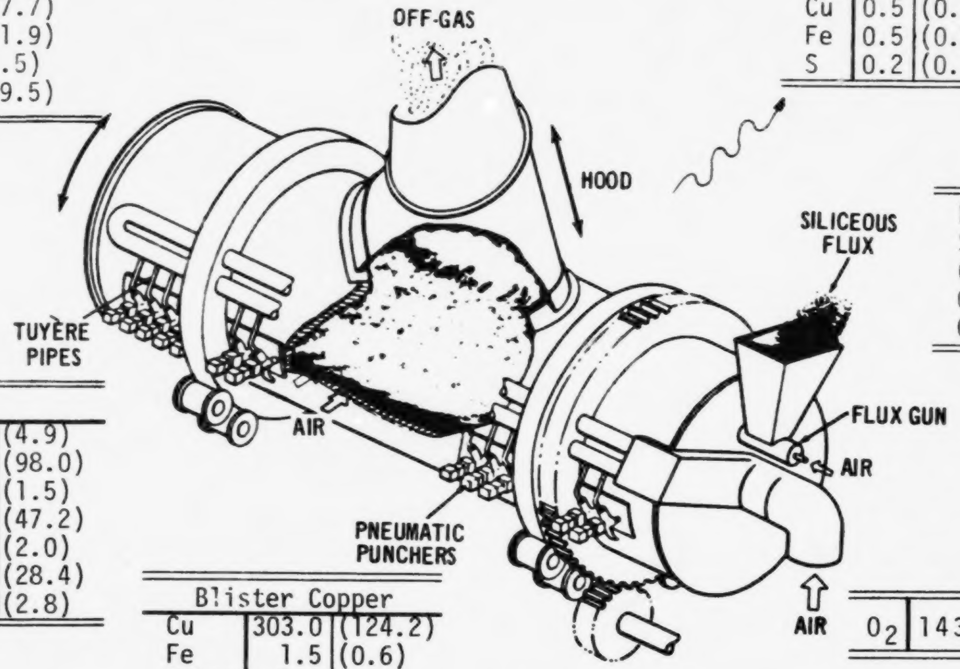
Figure 3. Reverberatory smelting furnace.*

*Source: Background Information; EPA 450/2-74-002a, October 1974.

Matte		
Cu	315.5	(129.3)
Fe	238.5	(97.7)
S	126.6	(51.9)
CaO	3.7	(1.5)
O_2	47.7	(19.5)

SO_2	237.0	(97.2)	4% vol.

Fugitives		
SO_2	8.2	(3.4)
Cu	0.5	(0.2)
Fe	0.5	(0.2)
S	0.2	(0.1)


OFF-GAS
HOOD
SILICEOUS FLUX
TUYÈRE PIPES
AIR
FLUX GUN
AIR
PNEUMATIC PUNCHERS
AIR

Fe	2.6	(1.1)
SiO_2	115.2	(47.2)
CaO	1.3	(0.5)
O_2	0.7	(0.3)
Other	8.2	(3.4)

Slag		
Cu	12.0	(4.9)
Fe	239.1	(98.0)
S	3.7	(1.5)
SiO_2	115.2	(47.2)
CaO	5.0	(2.0)
O_2	69.2	(28.4)
Other	6.9	(2.8)

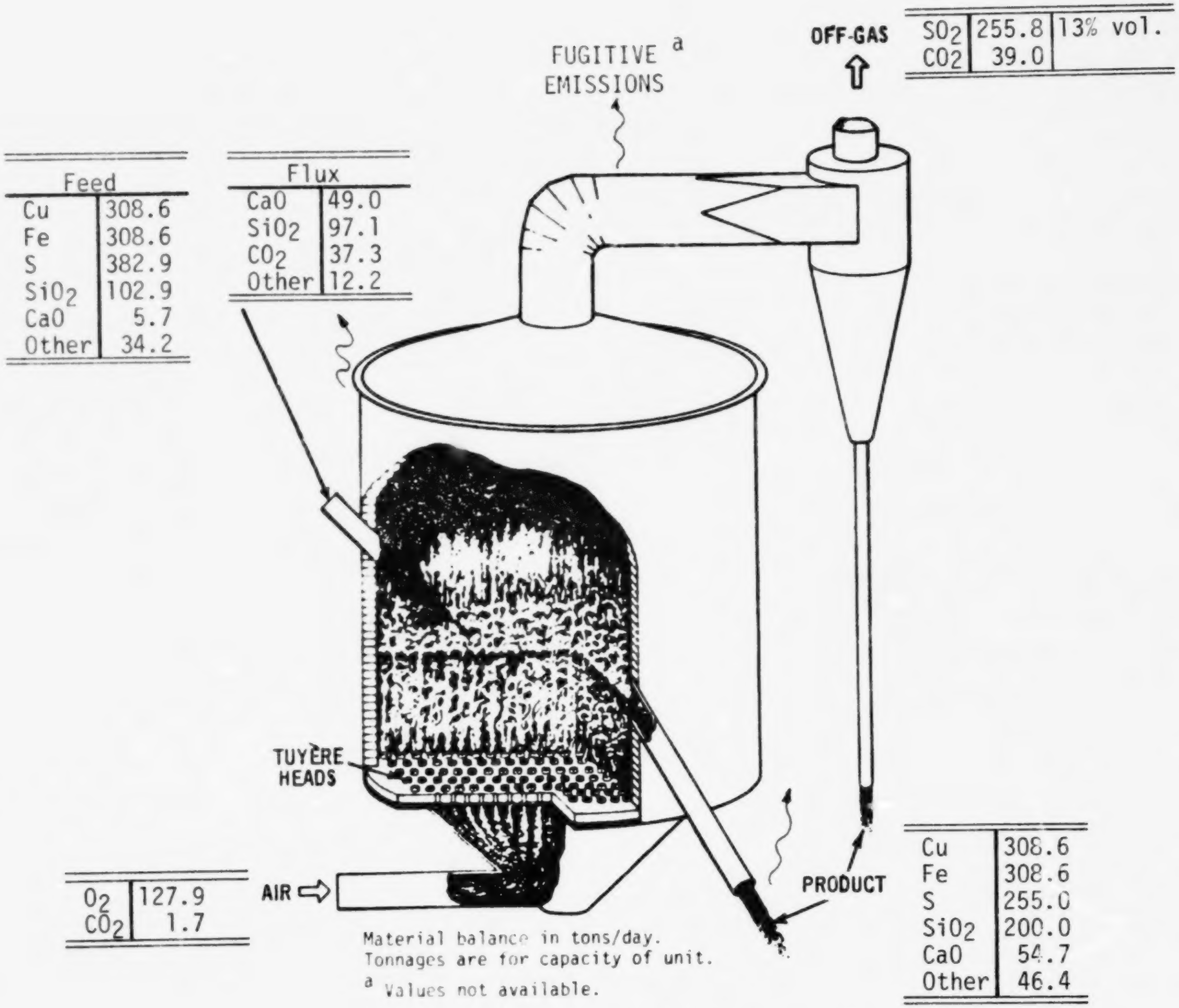
Blister Copper		
Cu	303.0	(124.2)
Fe	1.5	(0.6)
Other	1.5	(0.6)

O_2	143.3	(58.9)

Material balance in tons/day.
Values in parentheses indicate capacity of unit; other values are on basis of 303 tons/day blister copper.

Figure 4. Copper converter.*

*Source: Background Information; EPA 450/2-74-002a, October 1974.


FUGITIVE EMISSIONS[a]
OFF-GAS
TUYERE HEADS
AIR
PRODUCT

Feed	
Cu	308.6
Fe	308.6
S	382.9
SiO_2	102.9
CaO	5.7
Other	34.2

Flux	
CaO	49.0
SiO_2	97.1
CO_2	37.3
Other	12.2

SO_2	255.8	13% vol.
CO_2	39.0	

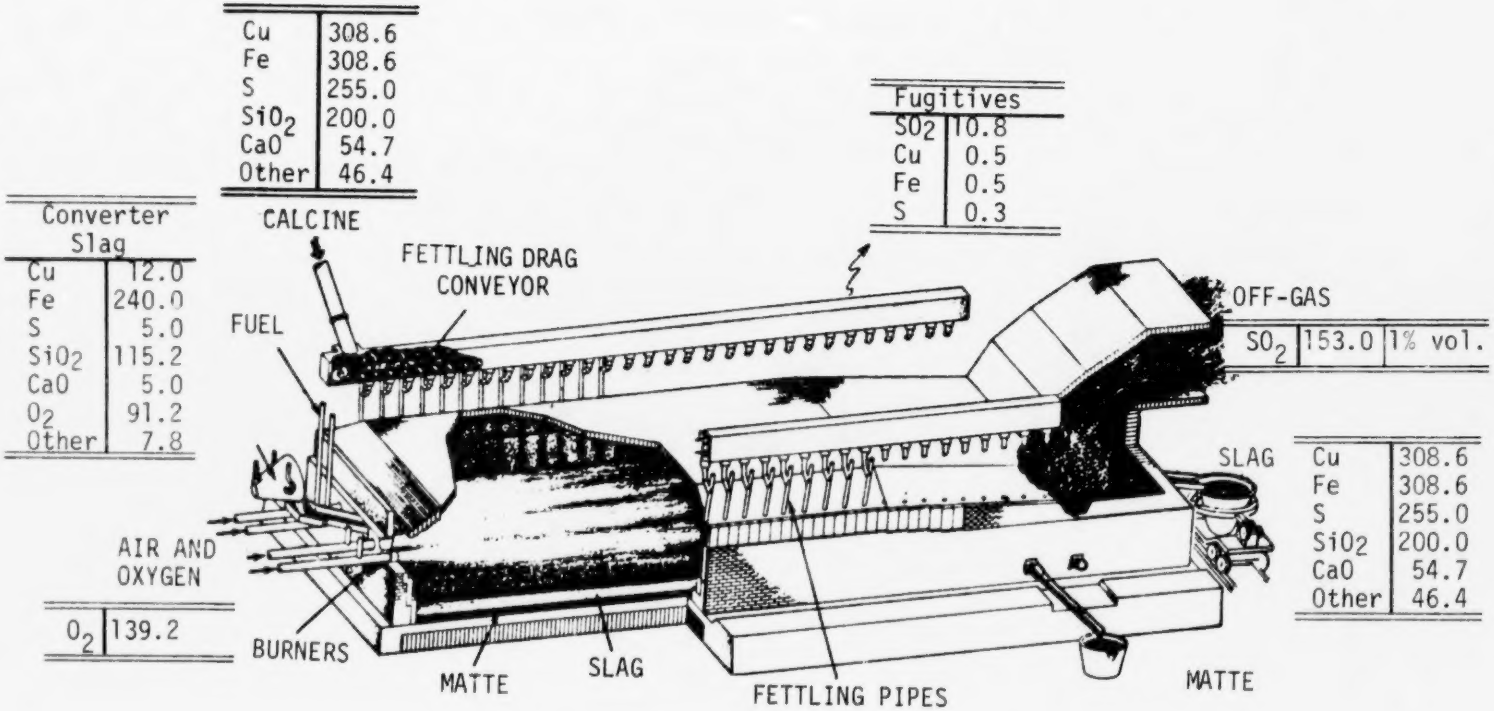
O_2	127.9
CO_2	1.7

Cu	308.6
Fe	308.6
S	255.0
SiO_2	200.0
CaO	54.7
Other	46.4

Material balance in tons/day.
Tonnages are for capacity of unit.
[a] Values not available.

Figure 5. Fluid-bed roaster.*

*Source: Background Information; EPA 450/2-74-002a, October 1974.


Cu 308.6
Fe 308.6
S 255.0
SiO_2 200.0
CaO 54.7
Other 46.4
CALCINE
Fugitives
SO_2 10.8
Cu 0.5
Fe 0.5
S 0.3
Converter Slag
Cu 12.0
Fe 240.0
S 5.0
SiO_2 115.2
CaO 5.0
O_2 91.2
Other 7.8
FETTLING DRAG CONVEYOR
FUEL
OFF-GAS
SO_2 153.0 1% vol.
SLAG
Cu 308.6
Fe 308.6
S 255.0
SiO_2 200.0
CaO 54.7
Other 46.4
AIR AND OXYGEN
O_2 139.2
BURNERS
MATTE
SLAG
FETTLING PIPES
MATTE

Material balance in tons/day.
Tonnages are for capacity of unit.

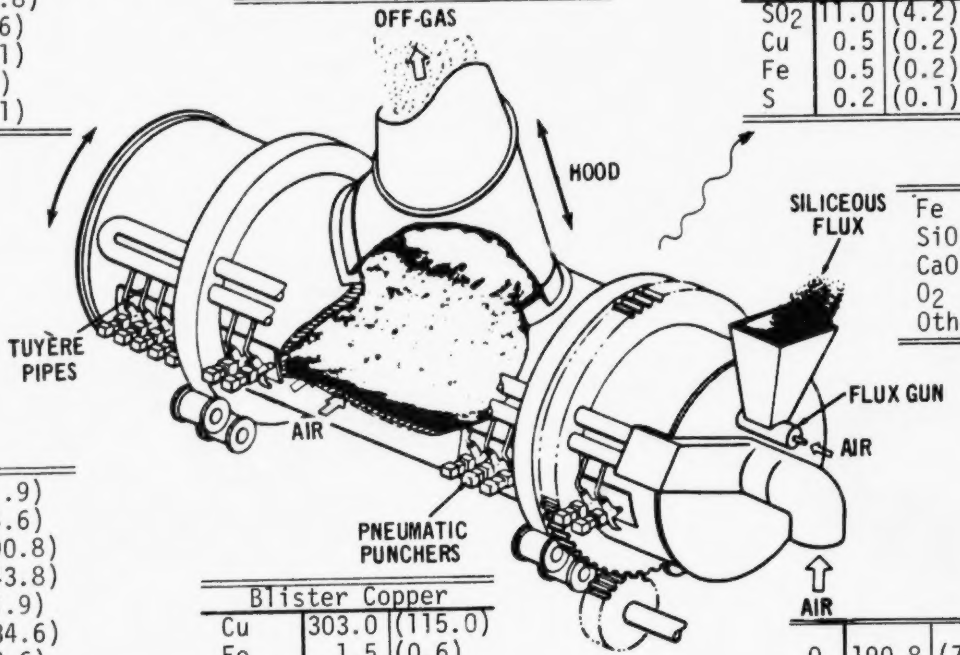
Figure 6. Reverberatory smelting furnace.*

*Source: Background Information; EPA 450/2-74-002a, October 1974.

Matte		
Cu	315.5	(119.8)
Fe	238.5	(90.6)
S	168.7	(64.1)
CaO	3.7	(1.4)
O_2	63.4	(24.1)

SO_2	316.0	(120.0)	4% vol.

Fugitives		
SO_2	11.0	(4.2)
Cu	0.5	(0.2)
Fe	0.5	(0.2)
S	0.2	(0.1)


OFF-GAS
HOOD
SILICEOUS FLUX
FLUX GUN
AIR
TUYÈRE PIPES
AIR
PNEUMATIC PUNCHERS
AIR

Fe	2.6	(1.0)
SiO_2	115.2	(43.8)
CaO	1.3	(0.5)
O_2	0.7	(0.3)
Other	8.2	(3.2)

SLAG		
S	5.0	(1.9)
Cu	12.0	(4.6)
Fe	239.1	(90.8)
SiO_2	115.2	(43.8)
CaO	5.0	(1.9)
O_2	91.2	(34.6)
Other	6.9	(2.6)

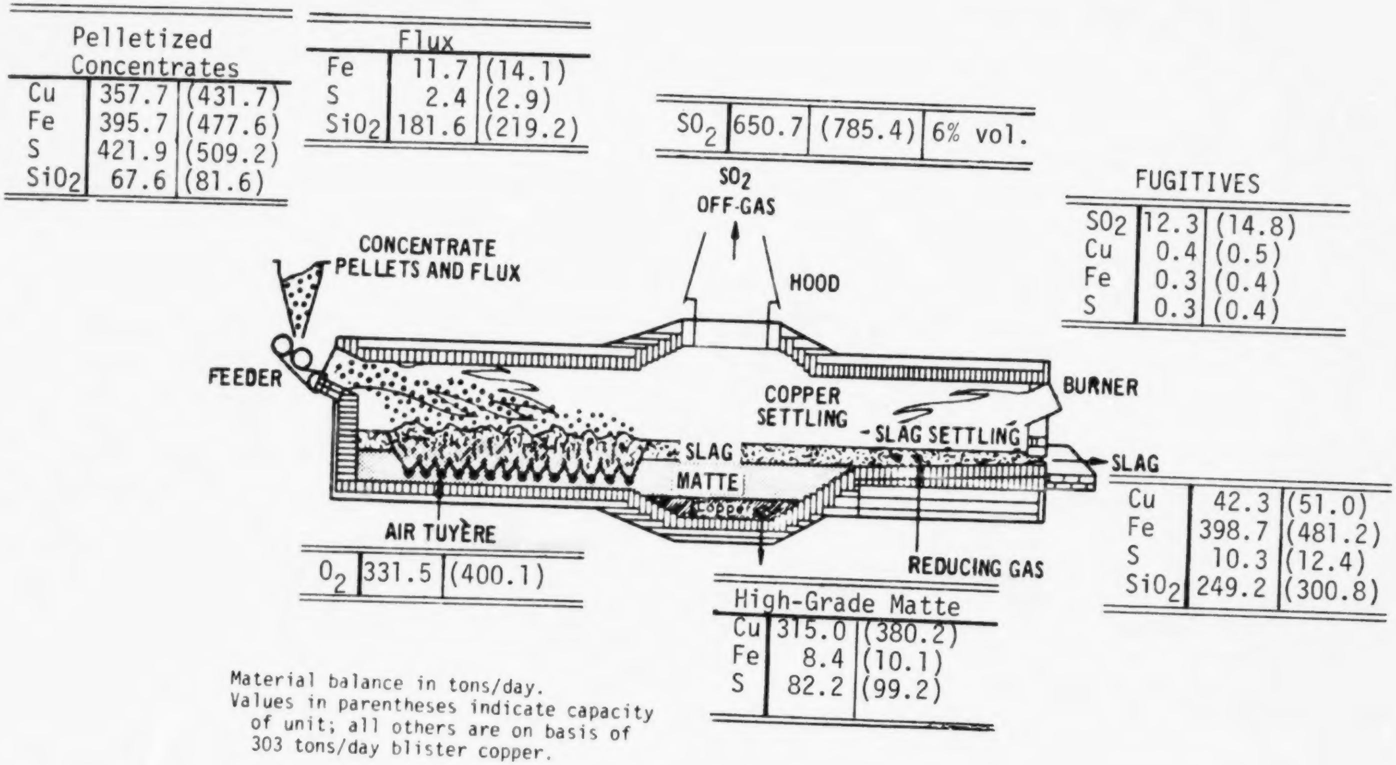
Blister Copper		
Cu	303.0	(115.0)
Fe	1.5	(0.6)
Other	1.5	(0.6)

O_2	190.8	(72.3)

Material balance in tons/day.
Values in parentheses are capacity of unit; all others are on basis of 303 tons/day blister copper.

Figure 7. Copper converter.*

*Source: Background Information; EPA 450/2-74-002a, October 1974.


CONCENTRATE PELLETS AND FLUX
FEEDER
AIR TUYÈRE
SLAG
MATTE
COPPER
COPPER SETTLING
SLAG SETTLING
SO2 OFF-GAS
HOOD
BURNER
SLAG
REDUCING GAS

Pelletized Concentrates		
Cu	357.7	(431.7)
Fe	395.7	(477.6)
S	421.9	(509.2)
SiO_2	67.6	(81.6)

Flux		
Fe	11.7	(14.1)
S	2.4	(2.9)
SiO_2	181.6	(219.2)

SO_2	650.7	(785.4)	6% vol.

FUGITIVES		
SO_2	12.3	(14.8)
Cu	0.4	(0.5)
Fe	0.3	(0.4)
S	0.3	(0.4)

O_2	331.5	(400.1)

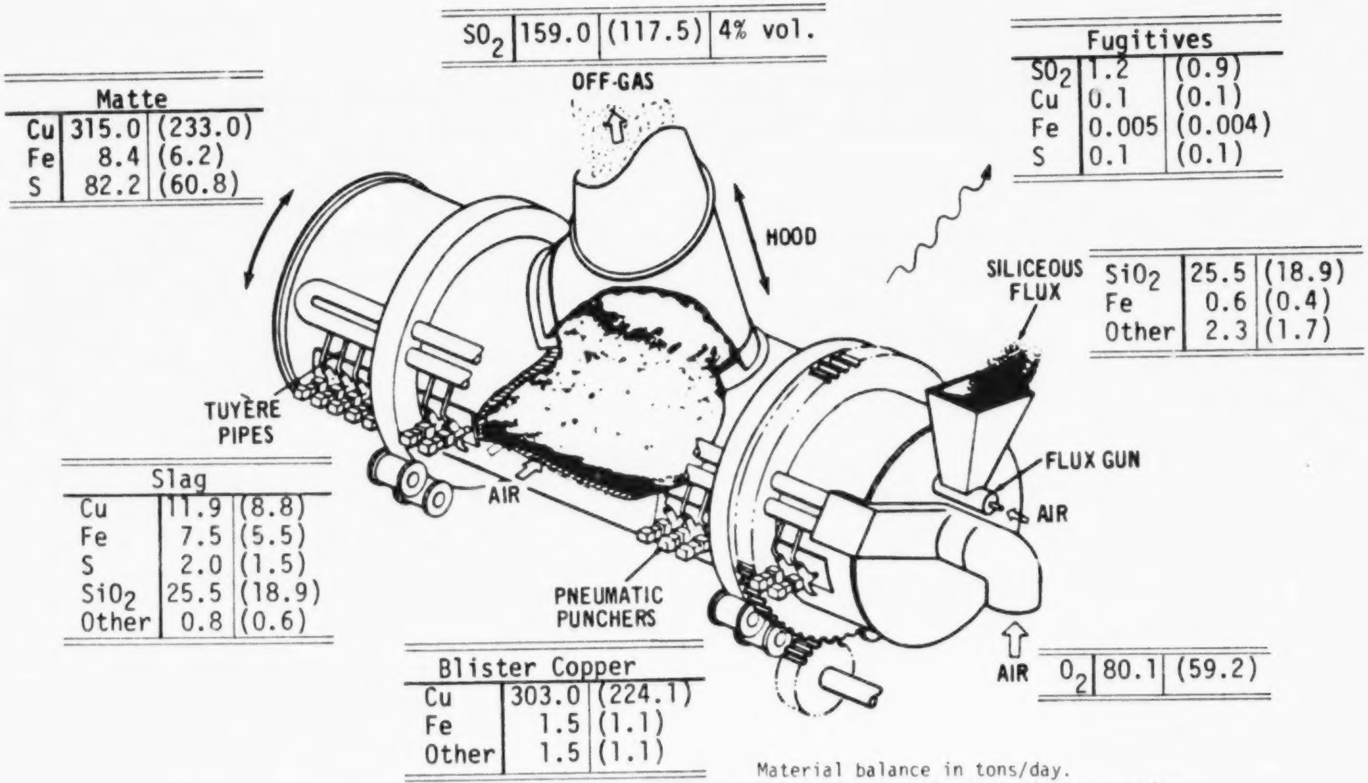
High-Grade Matte		
Cu	315.0	(380.2)
Fe	8.4	(10.1)
S	82.2	(99.2)

Cu	42.3	(51.0)
Fe	398.7	(481.2)
S	10.3	(12.4)
SiO_2	249.2	(300.8)

Material balance in tons/day.
Values in parentheses indicate capacity of unit; all others are on basis of 303 tons/day blister copper.

Figure 8. Noranda continuous smelting.*

*Source: Background Information; EPA 450/2-74-002a, October 1974.


OFF-GAS
HOOD
SILICEOUS FLUX
FLUX GUN
AIR
TUYÈRE PIPES
AIR
PNEUMATIC PUNCHERS
AIR

SO_2	159.0	(117.5)	4% vol.

Matte		
Cu	315.0	(233.0)
Fe	8.4	(6.2)
S	82.2	(60.8)

Fugitives		
SO_2	1.2	(0.9)
Cu	0.1	(0.1)
Fe	0.005	(0.004)
S	0.1	(0.1)

SiO_2	25.5	(18.9)
Fe	0.6	(0.4)
Other	2.3	(1.7)

Slag		
Cu	11.9	(8.8)
Fe	7.5	(5.5)
S	2.0	(1.5)
SiO_2	25.5	(18.9)
Other	0.8	(0.6)

Blister Copper		
Cu	303.0	(224.1)
Fe	1.5	(1.1)
Other	1.5	(1.1)

O_2	80.1	(59.2)

Material balance in tons/day.
Values in parentheses indicate capacity of unit; all others are on basis of 303 tons/day blister copper.

Figure 9. Copper converter.*

*Source: Background Information; EPA 450/2-74-002a, October 1974.

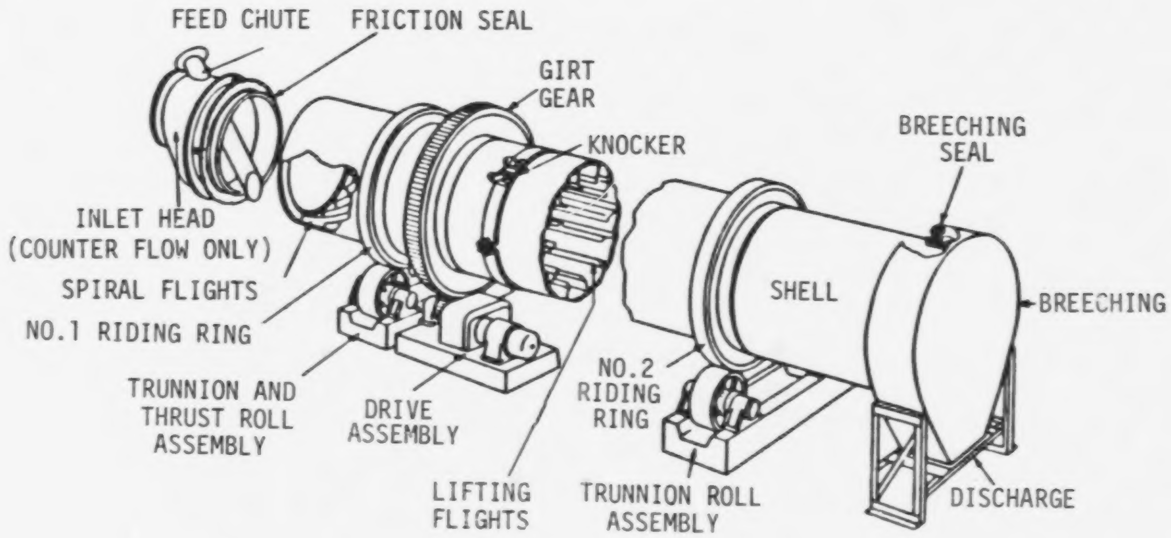

FEED CHUTE
FRICTION SEAL
GIRT GEAR
KNOCKER
BREECHING SEAL
INLET HEAD (COUNTER FLOW ONLY)
SPIRAL FLIGHTS
NO.1 RIDING RING
SHELL
BREECHING
TRUNNION AND THRUST ROLL ASSEMBLY
DRIVE ASSEMBLY
NO.2 RIDING RING
LIFTING FLIGHTS
TRUNNION ROLL ASSEMBLY
DISCHARGE

Figure 10. Pictorial of a countercurrent direct-heat rotary dryer.*
(Combustion chamber not shown.)

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* Source: Adapted from Surface Mining and Our Environment, U.S. Department of the Interior. 1967.

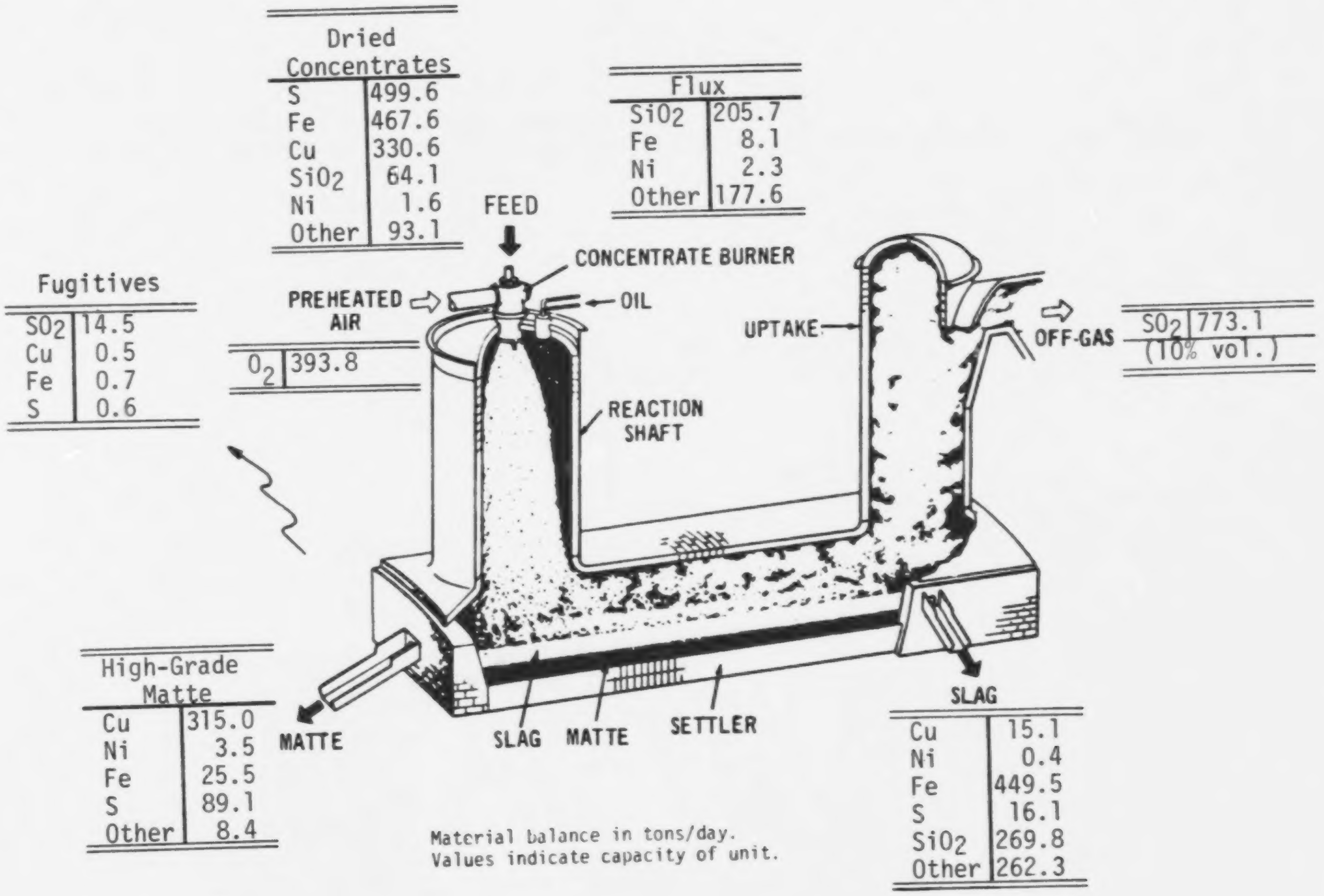
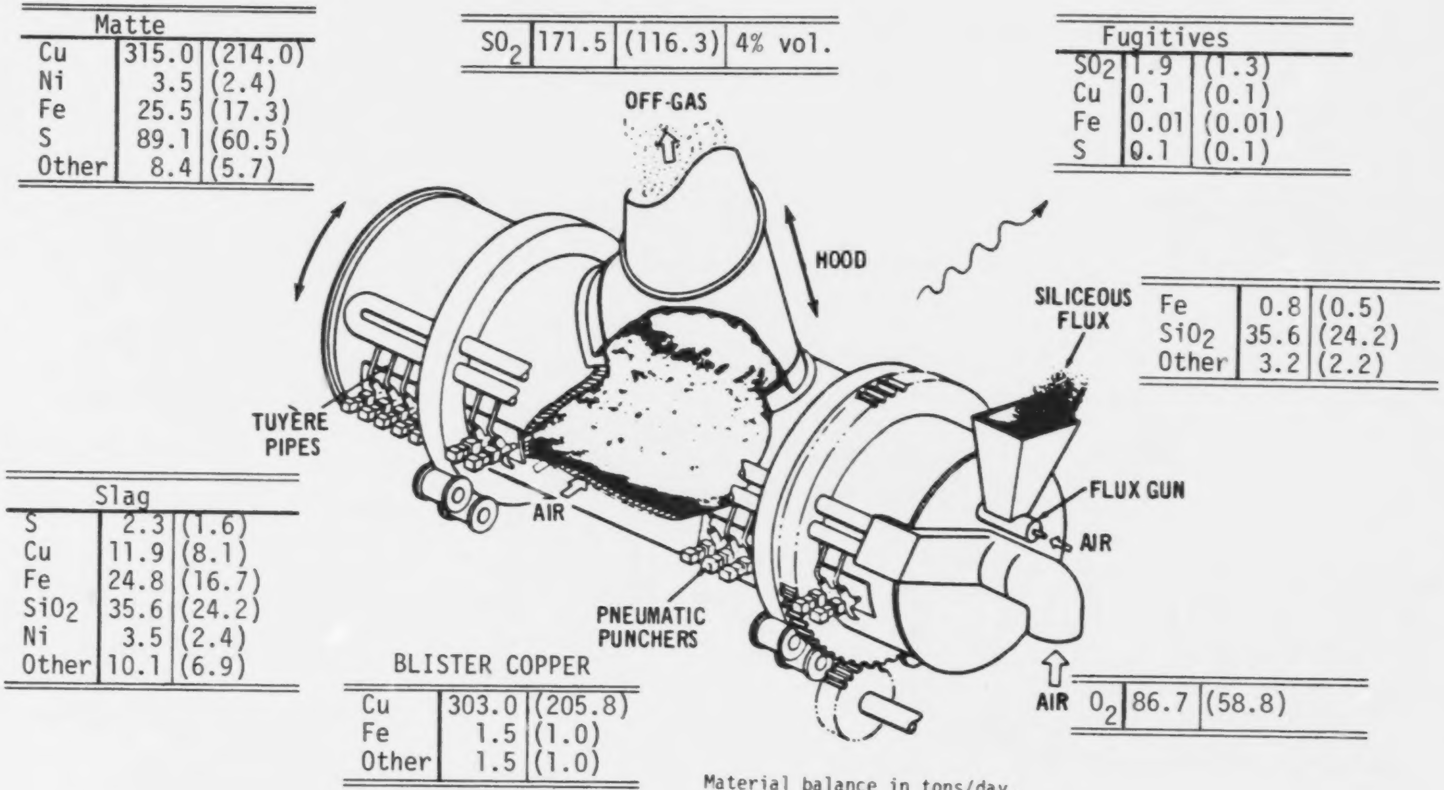

Dried Concentrates
S 499.6
Fe 467.6
Cu 330.6
SiO_2 64.1
Ni 1.6
Other 93.1
Flux
SiO_2 205.7
Fe 8.1
Ni 2.3
Other 177.6
FEED
CONCENTRATE BURNER
PREHEATED AIR
OIL
O_2 393.8
Fugitives
SO_2 14.5
Cu 0.5
Fe 0.7
S 0.6
UPTAKE
OFF-GAS
SO_2 773.1
(10% vol.)
REACTION SHAFT
High-Grade Matte
Cu 315.0
Ni 3.5
Fe 25.5
S 89.1
Other 8.4
MATTE
SLAG
MATTE
SETTLER
SLAG
Cu 15.1
Ni 0.4
Fe 449.5
S 16.1
SiO_2 269.8
Other 262.3
Material balance in tons/day.
Values indicate capacity of unit.

Figure 11. Outokumpu flash smelting furnace.*

*Source: Background Information; EPA 450/2-74-002a, October 1974.


OFF-GAS
HOOD
SILICEOUS FLUX
FLUX GUN
AIR
TUYÈRE PIPES
AIR
PNEUMATIC PUNCHERS
AIR

Matte		
Cu	315.0	(214.0)
Ni	3.5	(2.4)
Fe	25.5	(17.3)
S	89.1	(60.5)
Other	8.4	(5.7)

SO_2	171.5	(116.3)	4% vol.

Fugitives		
SO_2	1.9	(1.3)
Cu	0.1	(0.1)
Fe	0.01	(0.01)
S	0.1	(0.1)

Fe	0.8	(0.5)
SiO_2	35.6	(24.2)
Other	3.2	(2.2)

Slag		
S	2.3	(1.6)
Cu	11.9	(8.1)
Fe	24.8	(16.7)
SiO_2	35.6	(24.2)
Ni	3.5	(2.4)
Other	10.1	(6.9)

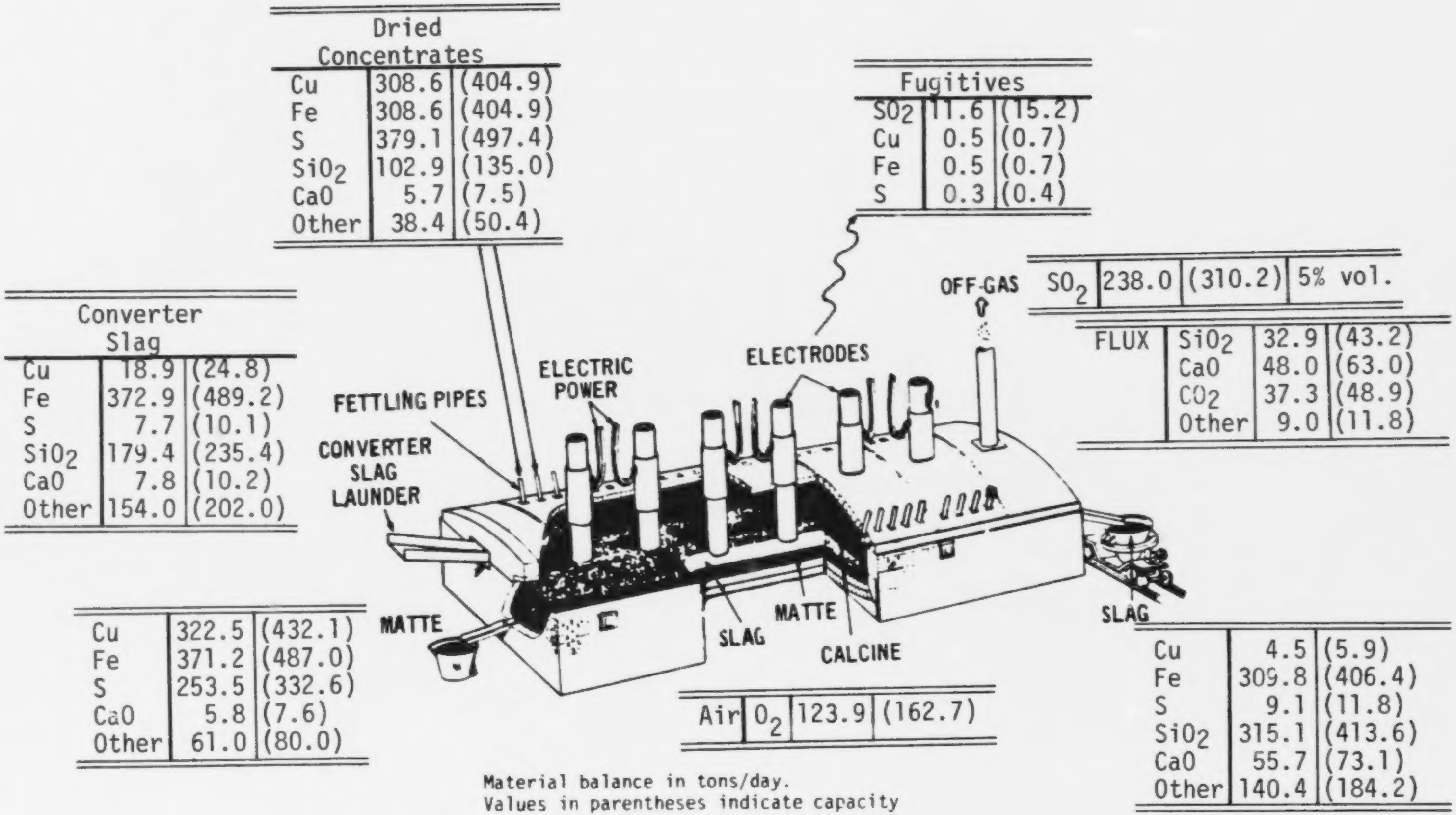
BLISTER COPPER		
Cu	303.0	(205.8)
Fe	1.5	(1.0)
Other	1.5	(1.0)

O_2	86.7	(58.8)

Material balance in tons/day.
Values in parentheses indicate capacity of unit; all other values are based on 303 tons/day blister copper.

Figure 12. Copper converter.*

*Source: Background Information; EPA 450/2-74-002a, October 1974.


FETTLING PIPES
CONVERTER SLAG LAUNDER
ELECTRIC POWER
ELECTRODES
OFF-GAS
MATTE
SLAG
MATTE
CALCINE
SLAG

Dried Concentrates		
Cu	308.6	(404.9)
Fe	308.6	(404.9)
S	379.1	(497.4)
SiO_2	102.9	(135.0)
CaO	5.7	(7.5)
Other	38.4	(50.4)

Fugitives		
SO_2	11.6	(15.2)
Cu	0.5	(0.7)
Fe	0.5	(0.7)
S	0.3	(0.4)

Converter Slag		
Cu	18.9	(24.8)
Fe	372.9	(489.2)
S	7.7	(10.1)
SiO_2	179.4	(235.4)
CaO	7.8	(10.2)
Other	154.0	(202.0)

SO_2	238.0	(310.2)	5% vol.

FLUX	SiO_2	32.9	(43.2)
	CaO	48.0	(63.0)
	CO_2	37.3	(48.9)
	Other	9.0	(11.8)

Cu	322.5	(432.1)
Fe	371.2	(487.0)
S	253.5	(332.6)
CaO	5.8	(7.6)
Other	61.0	(80.0)

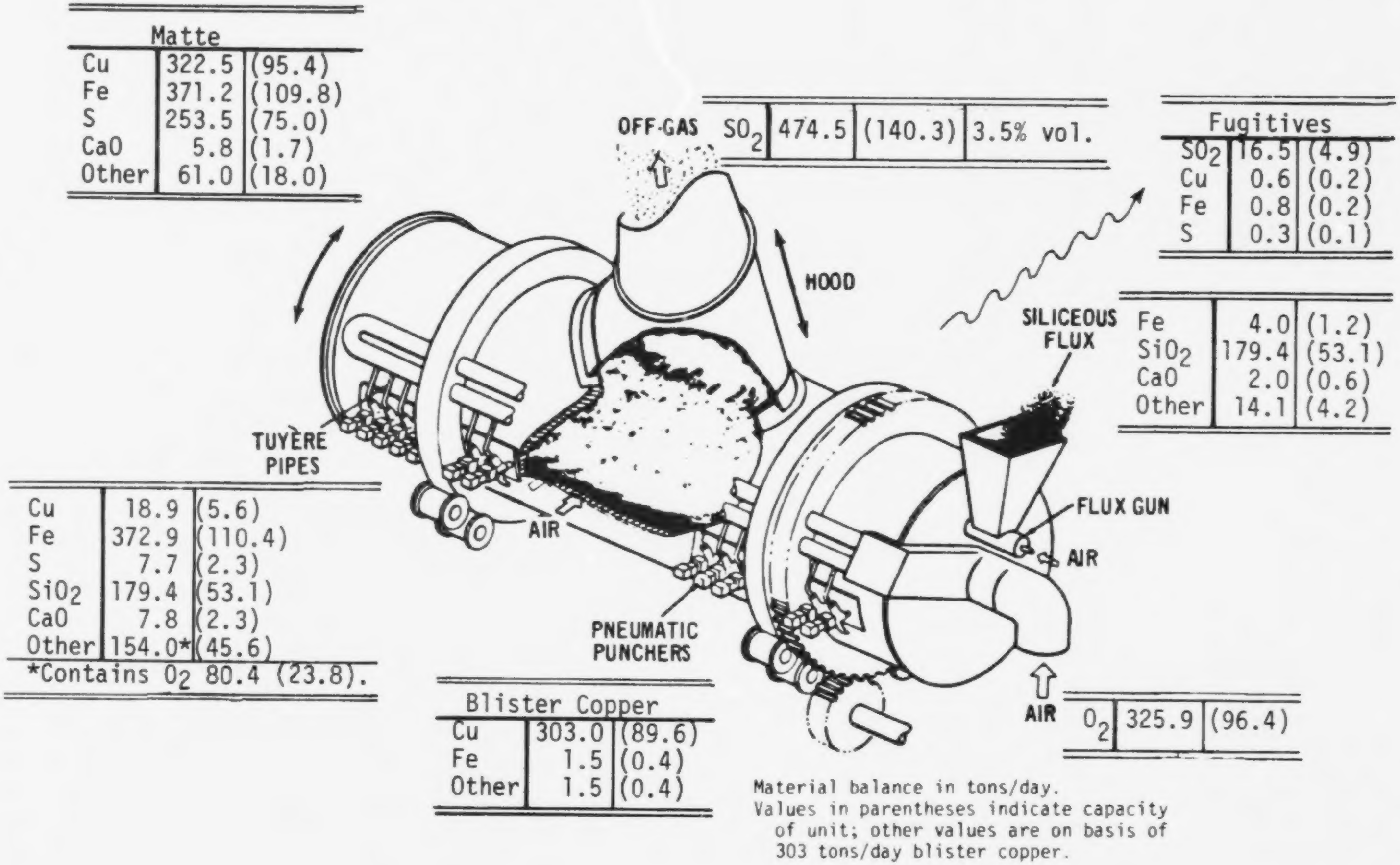
Air	O_2	123.9	(162.7)

Cu	4.5	(5.9)
Fe	309.8	(406.4)
S	9.1	(11.8)
SiO_2	315.1	(413.6)
CaO	55.7	(73.1)
Other	140.4	(184.2)

Material balance in tons/day.
Values in parentheses indicate capacity of unit; other values are on basis of 303 tons/day blister copper.

Figure 13. Electric smelting furnace.*

*Source: Background Information; EPA 450/2-74-002a, October 1974.


OFF-GAS
HOOD
SILICEOUS FLUX
FLUX GUN
AIR
TUYERE PIPES
AIR
PNEUMATIC PUNCHERS
AIR

Matte		
Cu	322.5	(95.4)
Fe	371.2	(109.8)
S	253.5	(75.0)
CaO	5.8	(1.7)
Other	61.0	(18.0)

OFF-GAS			
SO_2	474.5	(140.3)	3.5% vol.

Fugitives		
SO_2	16.5	(4.9)
Cu	0.6	(0.2)
Fe	0.8	(0.2)
S	0.3	(0.1)

Fe	4.0	(1.2)
SiO_2	179.4	(53.1)
CaO	2.0	(0.6)
Other	14.1	(4.2)

Cu	18.9	(5.6)
Fe	372.9	(110.4)
S	7.7	(2.3)
SiO_2	179.4	(53.1)
CaO	7.8	(2.3)
Other	154.0*	(45.6)

*Contains O_2 80.4 (23.8).

Blister Copper		
Cu	303.0	(89.6)
Fe	1.5	(0.4)
Other	1.5	(0.4)

AIR		
O_2	325.9	(96.4)

Material balance in tons/day.
Values in parentheses indicate capacity of unit; other values are on basis of 303 tons/day blister copper.

Figure 14. Copper converter.*

*Source: Background Information; EPA 450/2-74-002a, October 1974.

After looking at the existing process equipment, we suggest an attempt be made to minimize fugitive emissions from these processes by applying any of the following techniques where feasible. As an illustration on most operations, increasing the available draft for process gases reduces the fugitives but also dilutes the gas stream's SO_2 content, increases the size of all following equipment, etc. (We realize, of course, that each particular smelter has its own unique problems.)

ROASTERS

- Preventive maintenance to keep the unit tight and minimize leakage through the walls
- A fugitive collection system to collect emissions during the filling of a transfer or larry car from the roaster area
- A cylindrical envelope installed about the roaster. (Sufficient space should be left between the envelope and the roaster unit to allow maintenance; this space would be connected to a fugitive emission collection system.)

LARRY OR TRANSPORT CAR

- A collection system for fugitive emissions at both charging and discharging points
- A cover over the car when it is in transit
- Use of a pneumatic system for conveying the calcine product to the furnace storage bins

REVERBERATORY FURNACE

- Fugitive emission collection system, consisting of hooding and envelopes, for the launders and ladles
- Preventive maintenance on refractories, etc. to minimize fugitive emissions
- Converter slag return opening fitted with a pivoted and method of slag return

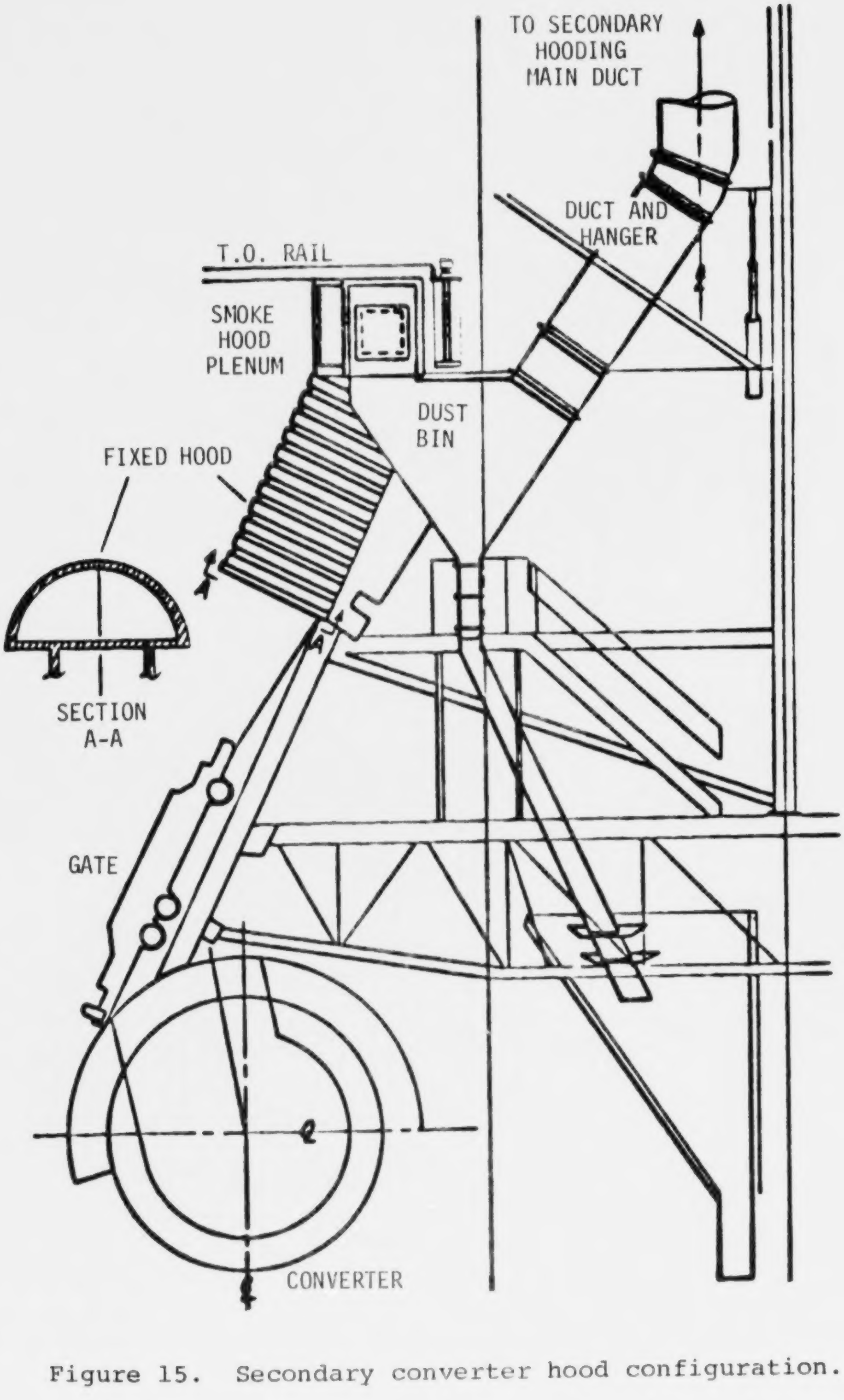

TO SECONDARY
HOODING
MAIN DUCT
DUCT AND
HANGER
T.O. RAIL
SMOKE
HOOD
PLENUM
DUST
BIN
FIXED HOOD
A
A
SECTION
A-A
GATE
℄
℄
CONVERTER

Figure 15. Secondary converter hood configuration.

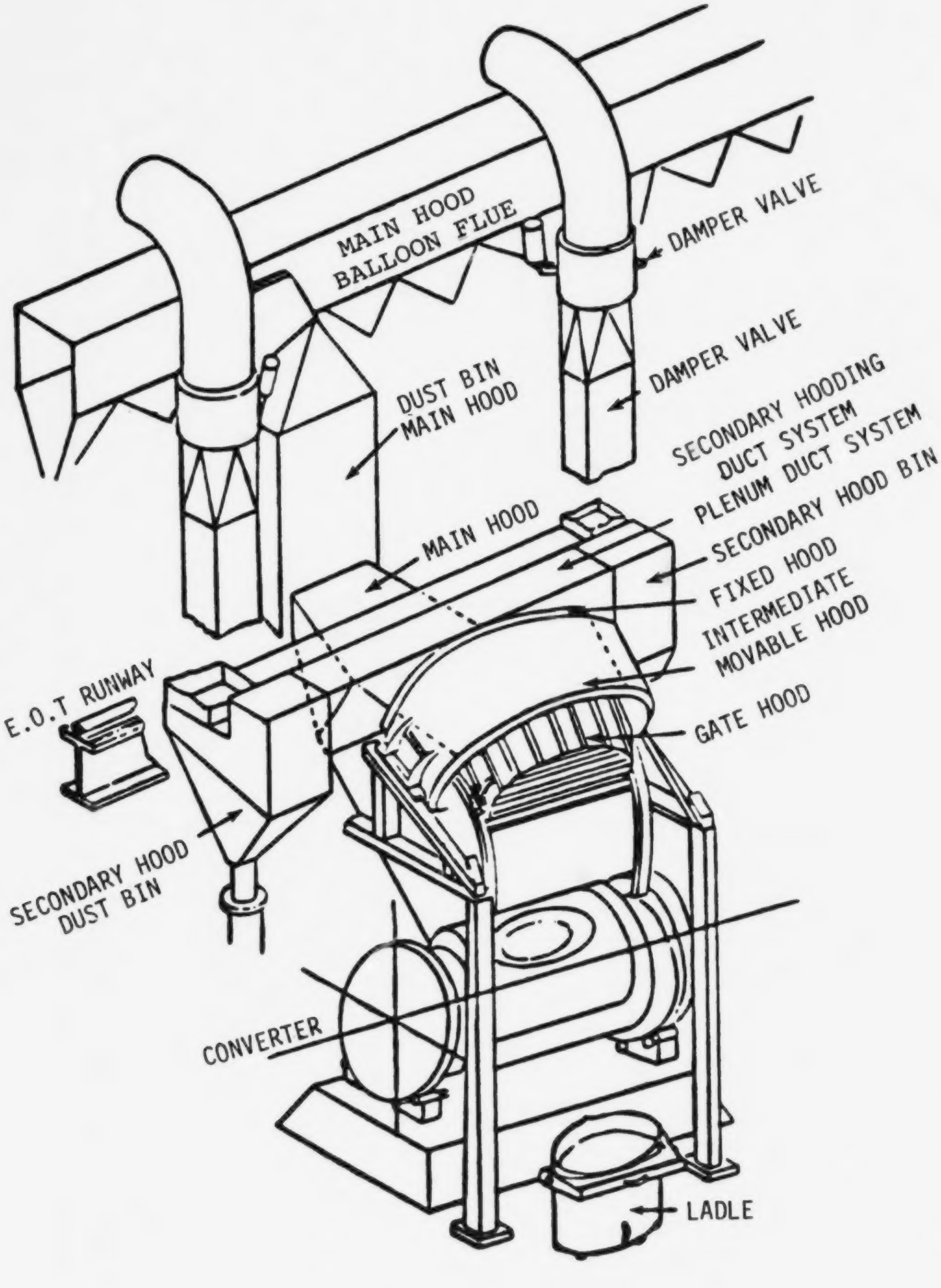
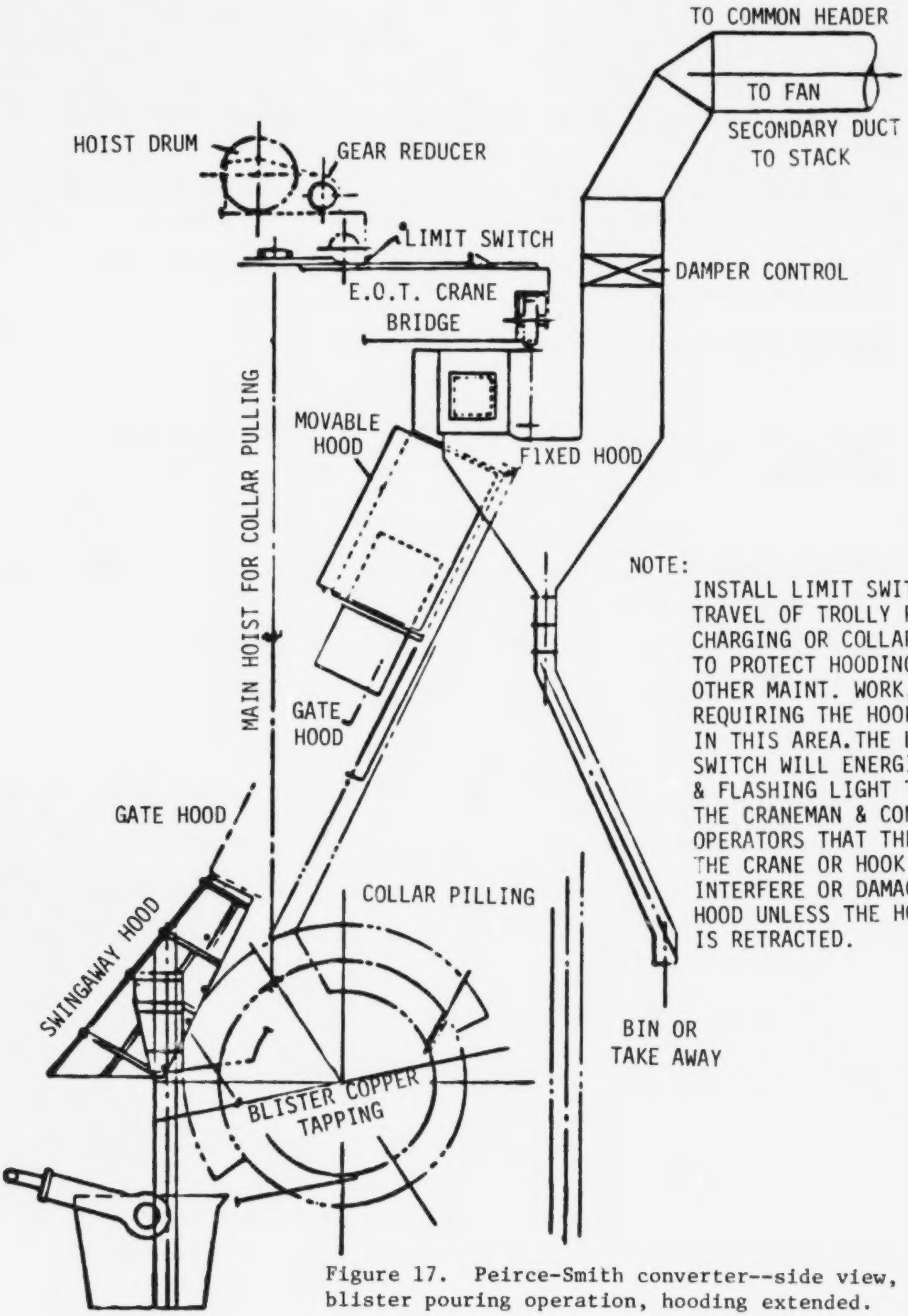

MAIN HOOD
BALLOON FLUE
DAMPER VALVE
DAMPER VALVE
DUST BIN
MAIN HOOD
SECONDARY HOODING
DUCT SYSTEM
PLENUM DUCT SYSTEM
SECONDARY HOOD BIN
MAIN HOOD
FIXED HOOD
INTERMEDIATE
MOVABLE HOOD
E.O.T RUNWAY
GATE HOOD
SECONDARY HOOD
DUST BIN
CONVERTER
LADLE

Figure 16. Peirce-Smith converter--retracted hooding, pictorial view.


TO COMMON HEADER
TO FAN
SECONDARY DUCT TO STACK
HOIST DRUM
GEAR REDUCER
LIMIT SWITCH
DAMPER CONTROL
E.O.T. CRANE BRIDGE
MAIN HOIST FOR COLLAR PULLING
MOVABLE HOOD
FIXED HOOD
GATE HOOD
GATE HOOD
SWINGAWAY HOOD
COLLAR PILLING
BLISTER COPPER TAPPING
BIN OR TAKE AWAY

NOTE:

INSTALL LIMIT SWITCH TO TRAVEL OF TROLLY FOR CHARGING OR COLLAR PULLING TO PROTECT HOODING: FOR OTHER MAINT. WORK, ETC. REQUIRING THE HOOK TO WORK IN THIS AREA. THE LIMIT SWITCH WILL ENERGIZE A GONG & FLASHING LIGHT TO ALERT THE CRANEMAN & CONVERTER OPERATORS THAT THE LOAD ON THE CRANE OR HOOK CAN INTERFERE OR DAMAGE THE HOOD UNLESS THE HOODING IS RETRACTED.

Figure 17. Peirce-Smith converter--side view, blister pouring operation, hooding extended.

ELECTRIC ARC FURNACE

- Preventive maintenance
- Feed transfer points to the furnace tied into fugitive emissions system
- A fugitive emission collector at the intersection of the electrode to the arch roofline, to handle leakage or paste burning
- Fugitive emission collection system of hooding and envelopes for the launders and ladles
- Pivoted cover fitted on slag return opening

DRYER

- Fugitive emission envelopes at charging and discharging ends to handle upsets

OUTOKUMPU

- Fugitive emission collection system of hooding and envelopes for the launders and ladles
- Fugitive emission collector at the concentrate injection
- Preventive maintenance on refractories, etc.

NORANDA

- Preventive maintenance on refractories, etc.
- Fugitive emission collector at the concentrate injection
- Fugitive emission collection system of hooding and envelopes for the launders and ladles
- Secondary hooding on roll-out

PEIRCE-SMITH CONVERTERS

- Rework the main hood for off-gases, with a minimum clearance to the converter
- Fugitive emission system hooding, including (where possible) swing-away, movable, etc. (Figures 15-17)
- Use of an air curtain control technique for fugitives (Appendix C)

HOBOKEN CONVERTERS

- The addition of a swingaway-hood type fugitive emission collection system for slagging and blister copper operations would virtually free this unit of fugitive emissions (Figure 18)
- Maintenance of proper draft within the converter and good operating procedures

ANODE FURNACES

- Addition of a swingaway-type fugitive emissions collection system

MATTE, SLAG, AND BLISTER COPPER TRANSPORT

- Use of a covered ladle with a stopper rod for bottom pouring to minimize emissions during transit or the use of vermiculite or equivalent as a cover for matte to minimize fugitives in lieu of a fixed cover to the ladle in transport
- Adaption of a telescopic stiff leg to the electric overhead traveling crane (EOT) for evacuation of fugitive emissions during transport of matte, slag, or blister copper or when charging or pouring (Figure 19)

TOTAL BUILDING ENCLOSURE

- Good control on air movements into and from the building
- Fan capacity with sufficient acfm
- Building design modification

The passing of time and the resulting need to update or replace facilities to remain competitive make it necessary to plan ahead to minimize operating costs and meet possible mandated pollution controls in a specific industry or area.

It is likely that primary copper smelters will be required to meet a host of regulations limiting SO_2, arsenic, and trace metals emissions. What methods are available to control these emissions and still remain cost competitive? The future system for achieving this goal may be one of the following methods or some combination of them.

CASCADING GRAVITY FLOW

This system (Figure 20a, b, c) would begin with the feed being conveyed to storage bins and then to either multihearth or fluid-bed roasters or to a dryer. The partially calcined product is collected and sent to storage bins or silos and then conveyed as required to the smelter furnace (i.e., a reverberatory, electric arc, Noranda, or Outokumpu). Where necessary, the feed is

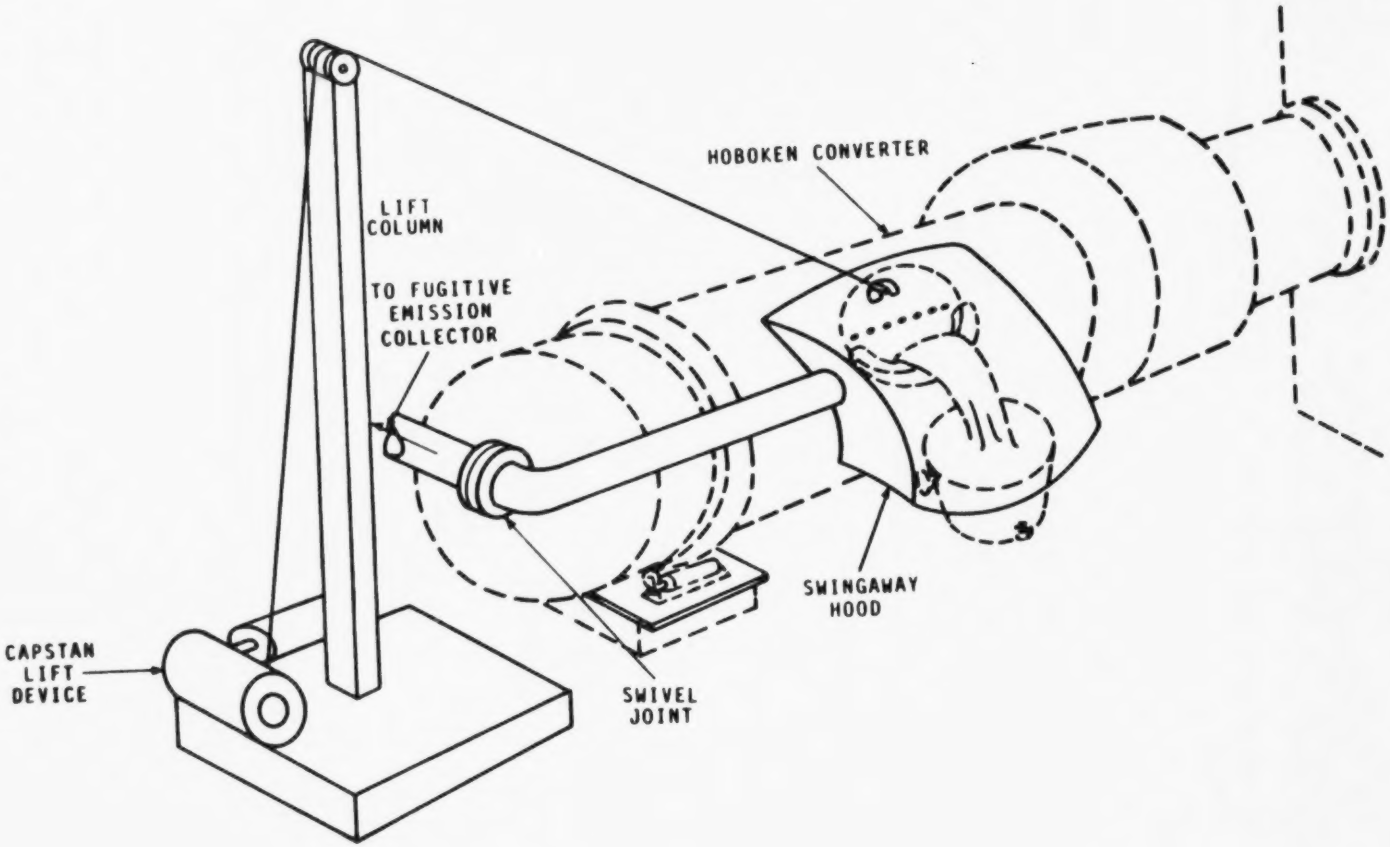

LIFT COLUMN
HOBOKEN CONVERTER
TO FUGITIVE EMISSION COLLECTOR
CAPSTAN LIFT DEVICE
SWINGAWAY HOOD
SWIVEL JOINT

Figure 18. Hoboken converter with swingaway hood.

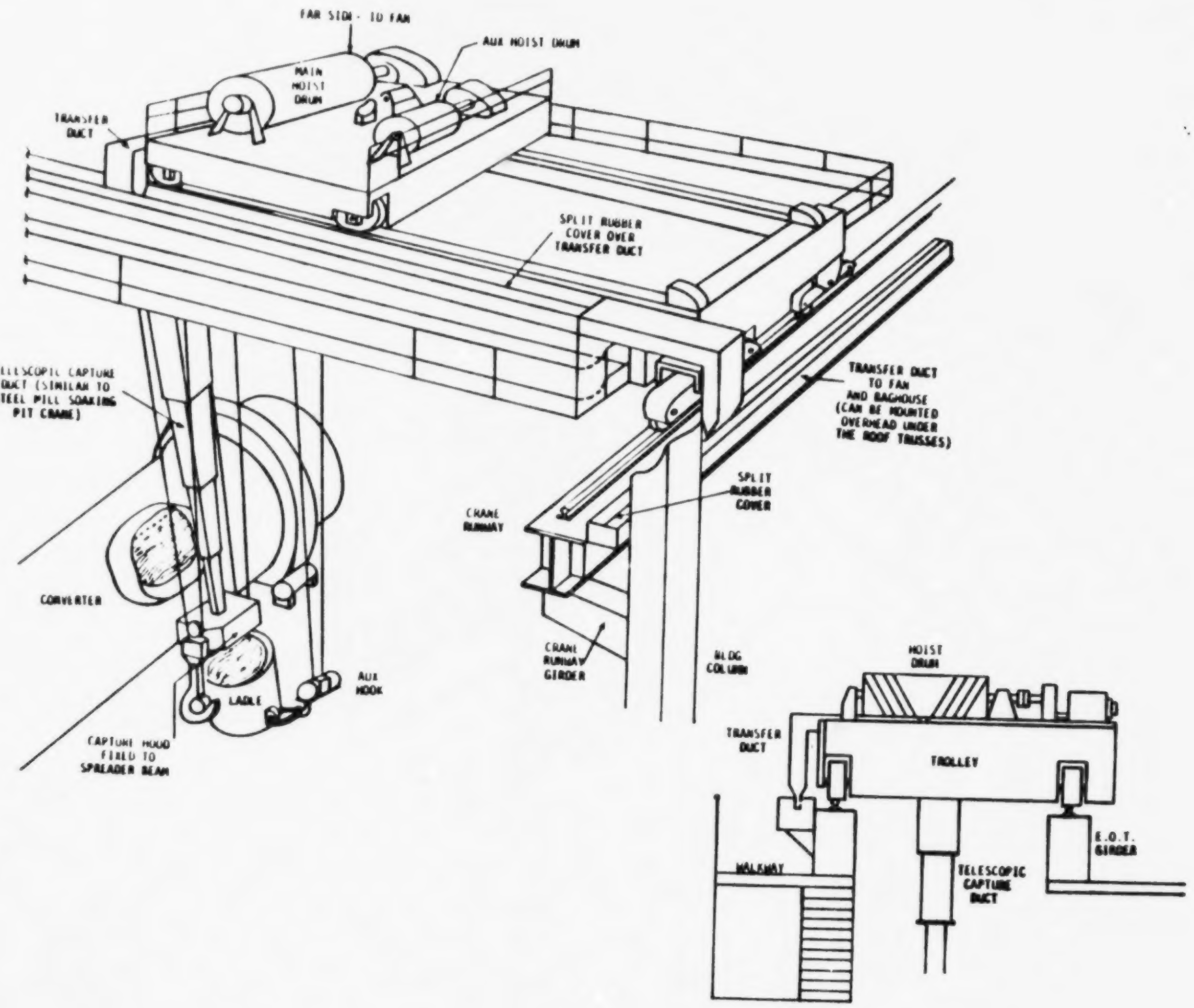

FAR SIDE - TO FAN
AUX HOIST DRUM
MAIN HOIST DRUM
TRANSFER DUCT
SPLIT RUBBER COVER OVER TRANSFER DUCT
TELESCOPIC CAPTURE DUCT (SIMILAR TO STEEL MILL SOAKING PIT CRANE)
TRANSFER DUCT TO FAN AND BAGHOUSE (CAN BE MOUNTED OVERHEAD UNDER THE ROOF TRUSSES)
SPLIT RUBBER COVER
CRANE RUNWAY
CONVERTER
CRANE RUNWAY GIRDER
BLDG COLUMN
HOIST DRUM
AUX HOOK
LADLE
CAPTURE HOOD FIXED TO SPREADER BEAM
TRANSFER DUCT
TROLLEY
E.O.T. GIRDER
WALKWAY
TELESCOPIC CAPTURE DUCT

Figure 19. Ladle/EOT crane fugitive emission collection system.

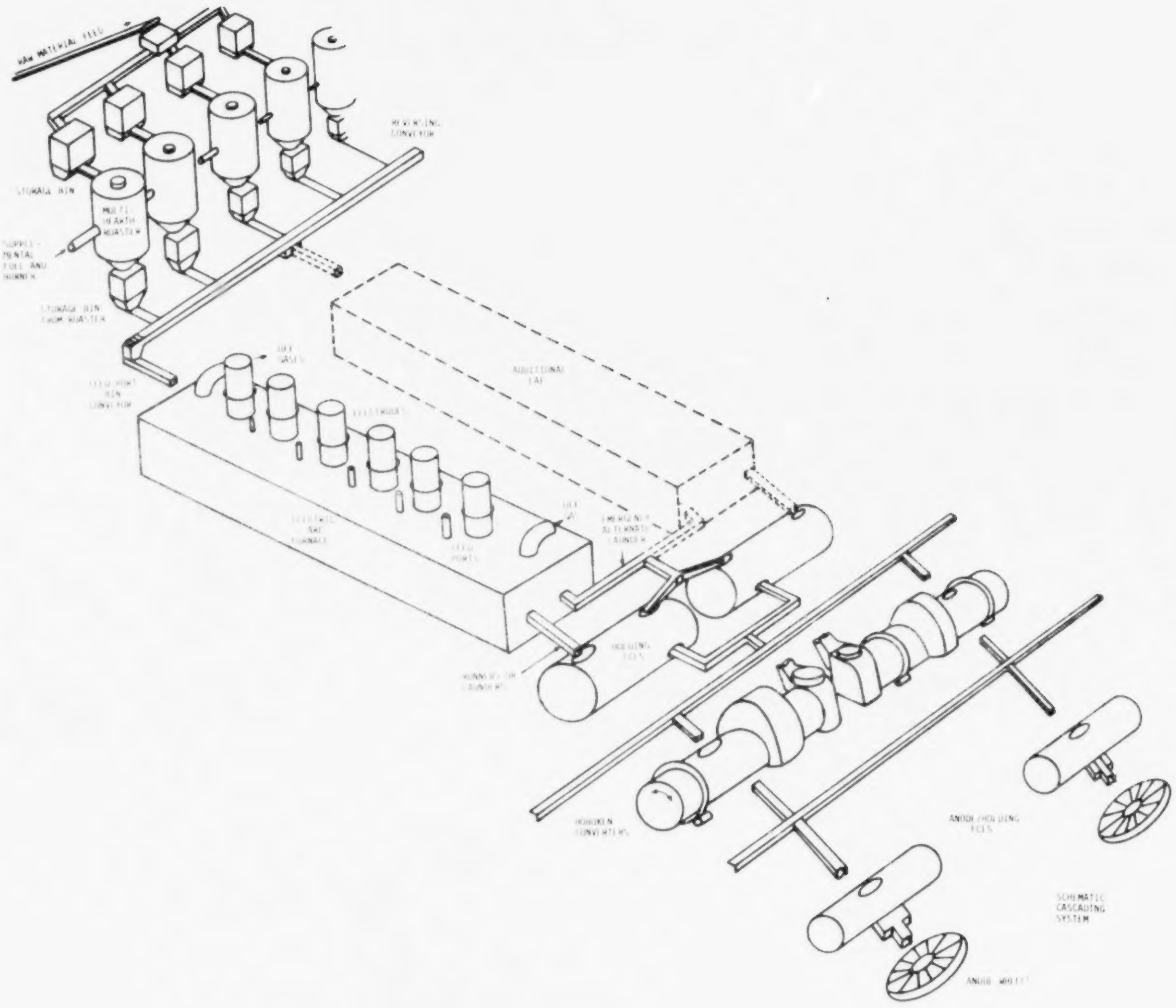

RAW MATERIAL FEED
REVERSING CONVEYOR
STORAGE BIN
MULTI-HEARTH ROASTER
SUPPLEMENTAL FUEL AND BURNER
STORAGE BIN FROM ROASTER
FEED PORT BIN CONVEYOR
OFF GASES
ELECTRODES
ADDITIONAL EAF
ELECTRIC ARC FURNACE
FEED PORTS
OFF GAS
EMERGENCY ALTERNATE LAUNDER
HOLDING FCES
RUNNERS OR LAUNDERS
HOBOKEN CONVERTERS
ANODE/HOLDING FCES
SCHEMATIC CASCADING SYSTEM
ANODE WHEELS

Figure 20a. Cascading gravity flow.

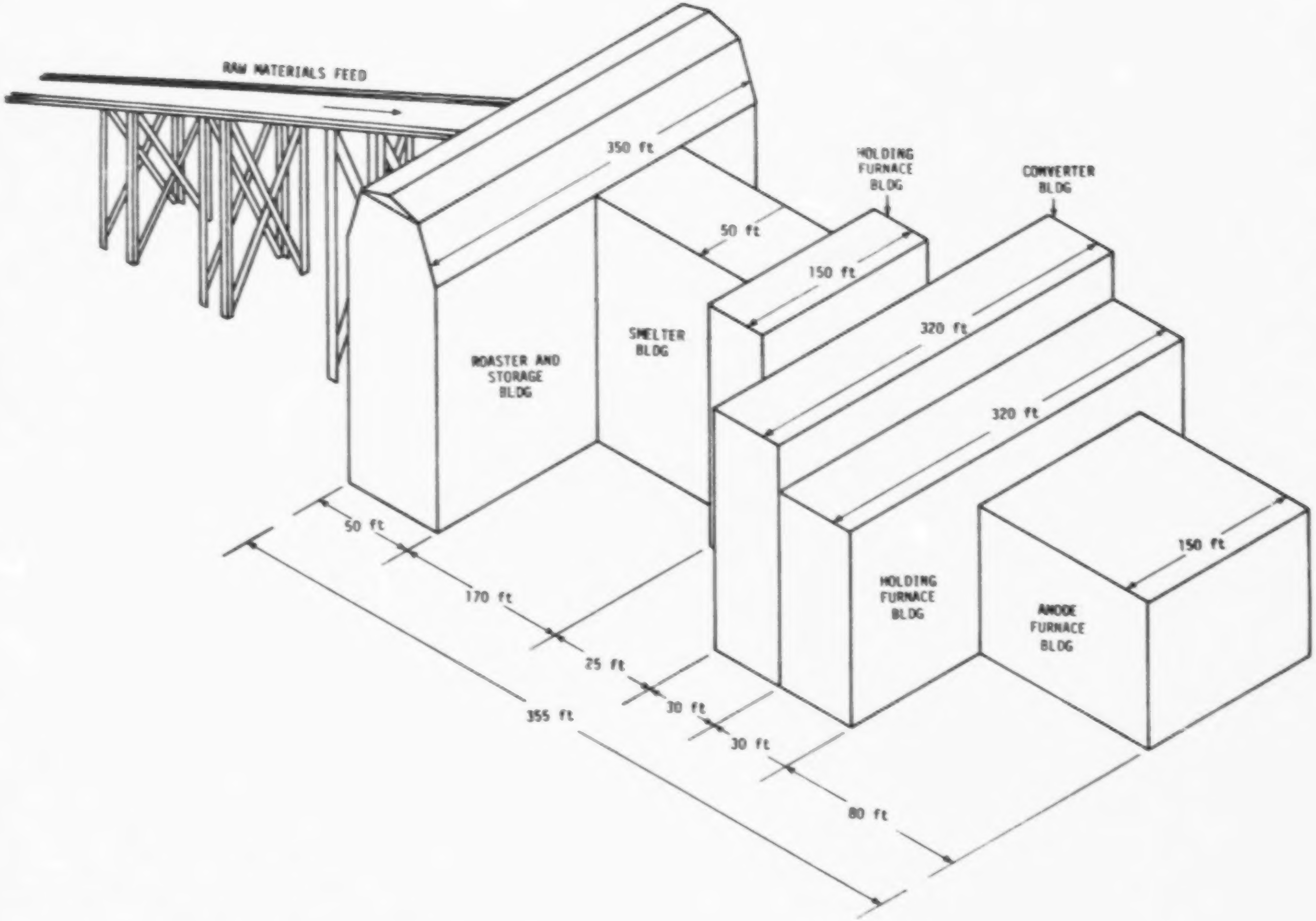

RAW MATERIALS FEED
350 ft
HOLDING FURNACE BLDG
CONVERTER BLDG
50 ft
150 ft
320 ft
320 ft
ROASTER AND STORAGE BLDG
SMELTER BLDG
HOLDING FURNACE BLDG
ANODE FURNACE BLDG
150 ft
50 ft
170 ft
25 ft
30 ft
30 ft
80 ft
355 ft

Figure 20b. Schematic cascading building outline.

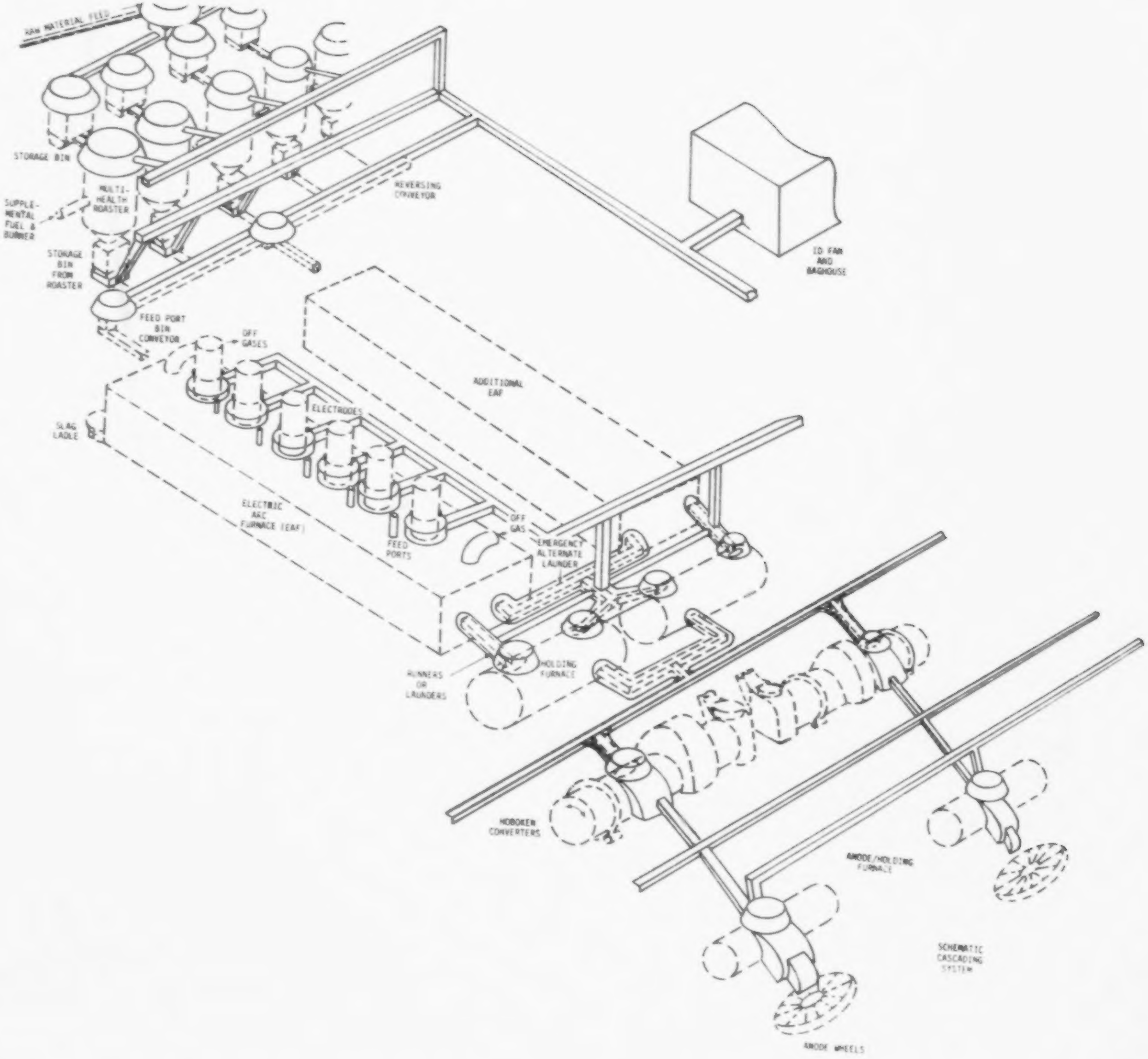

RAW MATERIAL FEED
STORAGE BIN
SUPPLE-MENTAL FUEL & BURNER
MULTI-HEALTH ROASTER
STORAGE BIN FROM ROASTER
REVERSING CONVEYOR
ID FAN AND BAGHOUSE
FEED PORT BIN CONVEYOR
OFF GASES
ADDITIONAL EAF
SLAG LADLE
ELECTRODES
ELECTRIC ARC FURNACE (EAF)
FEED PORTS
OFF GAS
EMERGENCY ALTERNATE LAUNDER
RUNNERS OR LAUNDERS
HOLDING FURNACE
HOBOKEN CONVERTERS
ANODE/HOLDING FURNACE
SCHEMATIC CASCADING SYSTEM
ANODE WHEELS

Figure 20c. Fugitive emission collection system for cascading gravity flow.

RELATIVE COSTS, EMISSIONS, AND ENERGY EVALUATION
EXISTING VERSUS ALTERNATE SYSTEMS

Cascading System Versus Conventional Primary Copper Smelter

Costs

Building costs may be greater.
Operational costs may be higher.

Emissions

Minimal

Energy Requirements

Higher for concentrate feed delivery because of height
Minimal for transporting from furnace to furnace

Pros and Cons

Preventive maintenance must be practiced.
Proximity of furnaces to each other
Change in operating techniques

prepared to the size required. After smelting, the matte flows by gravity, via launders, to the holding furnaces (induction-type or supplemental fuel-fired). From the holding furnaces, it flows by gravity, via launders, to Hoboken converters and then on to the anode furnace, or to holding furnaces and then to the anode casting wheels. The conventional transfer of the main process off-gases would be controlled to electrostatic precipitators or baghouses, which exhaust to an SO_2 recovery system (acid plant, liquid SO_2, etc.). The fugitive emissions (Figure 20b) would be ducted to one or more main collection systems via a baghouse prior to being discharged to the atmosphere. Collection would be made at all transfer points over launders and at the discharge to slag pots or ladles or holding furnaces. The vessel in which slag from the smelter is conveyed to a dump or refuse area would be capped or covered, prior to leaving the collection station. The Hoboken converter would have a swingaway hood to collect fugitive emissions during slagging or pouring. The slag ladle would be a stopper rod-type, fitted with a cover. During recharging to the smelter furnace, the ladle is positioned and mated to the furnace so that when the stopper rod is lifted the slag discharges into the furnace with minimal fugitive emissions.

CASCADING/INDUCTION/GRAVITY FLOW

This system (Figure 21a) would permit the use of conventional building heights, whereas the cascading gravity flow would require taller buildings. The roaster or dryers would be at ground level. The partially calcined feed would be conveyed to storage bins or silos and then to the smelting furnace. The matte would then be transported to the holding furnaces (Hobokens or anode furnaces) by means of induction coils that move the molten metal from a lower elevation to the higher level. The movement of the matte would be through refractory-lined pipe. Such an arrangement would minimize the need for fugitive emissions collection ducts over the launders to the holding furnaces. It would require the use of tundishes or covered troughs to convey the matte or blister copper to another furnace that is not in the direct line of discharge from one point to the next. Again, collection of fugitive emissions (Figure 21b) and handling of slag would be similar to that for the cascading gravity system.

SMELTING/CONVERTER FURNACES OF THE NORANDA OR OUTOKUMPU

With some slight modifications in operating and process techniques, it may be possible to bypass the holding and converter furnaces of the cascading system discussed above. Extending the hearth area (Figure 22) of the Outokumpu, using lances and supplemental feeder burners (i.e., pulverized feed and oil or coal), and adding another off-gas duct (similar to the Hoboken converter) for draft control might permit a continuous operation for discharging blister copper all in one vessel. Extending the Noranda center section (Figure 23), using lances in the area of the matte collection, and slightly increasing the off-gas hood size might enable delivery of blister copper in a one-step unit.

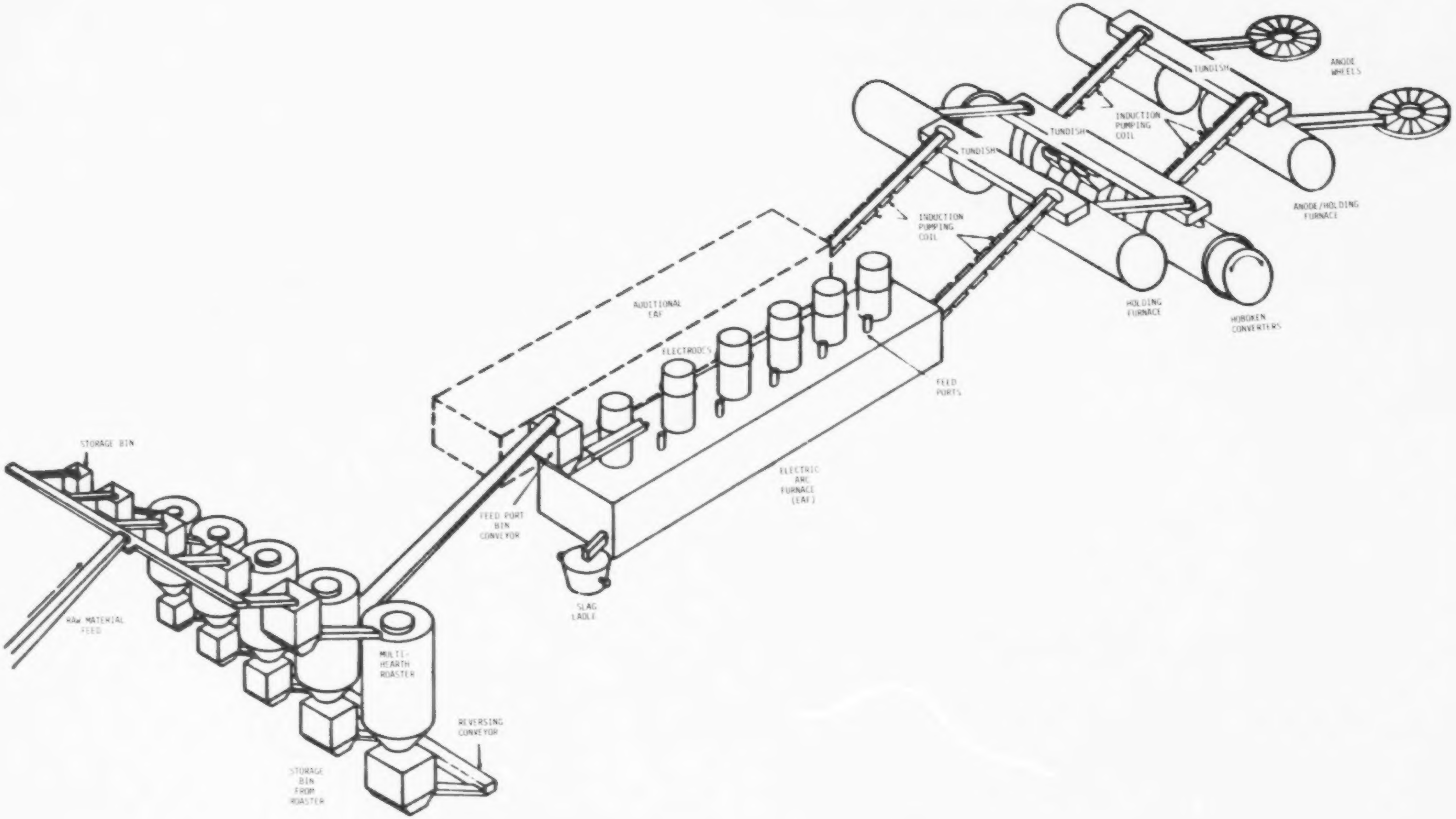

ANODE WHEELS
TUNDISH
INDUCTION PUMPING COIL
TUNDISH
TUNDISH
ANODE/HOLDING FURNACE
INDUCTION PUMPING COIL
HOLDING FURNACE
HOBOKEN CONVERTERS
ADDITIONAL EAF
ELECTRODES
FEED PORTS
STORAGE BIN
ELECTRIC ARC FURNACE (EAF)
FEED PORT BIN CONVEYOR
SLAG LADLE
RAW MATERIAL FEED
MULTI-HEARTH ROASTER
REVERSING CONVEYOR
STORAGE BIN FROM ROASTER

Figure 21a. Cascading/induction/gravity flow.

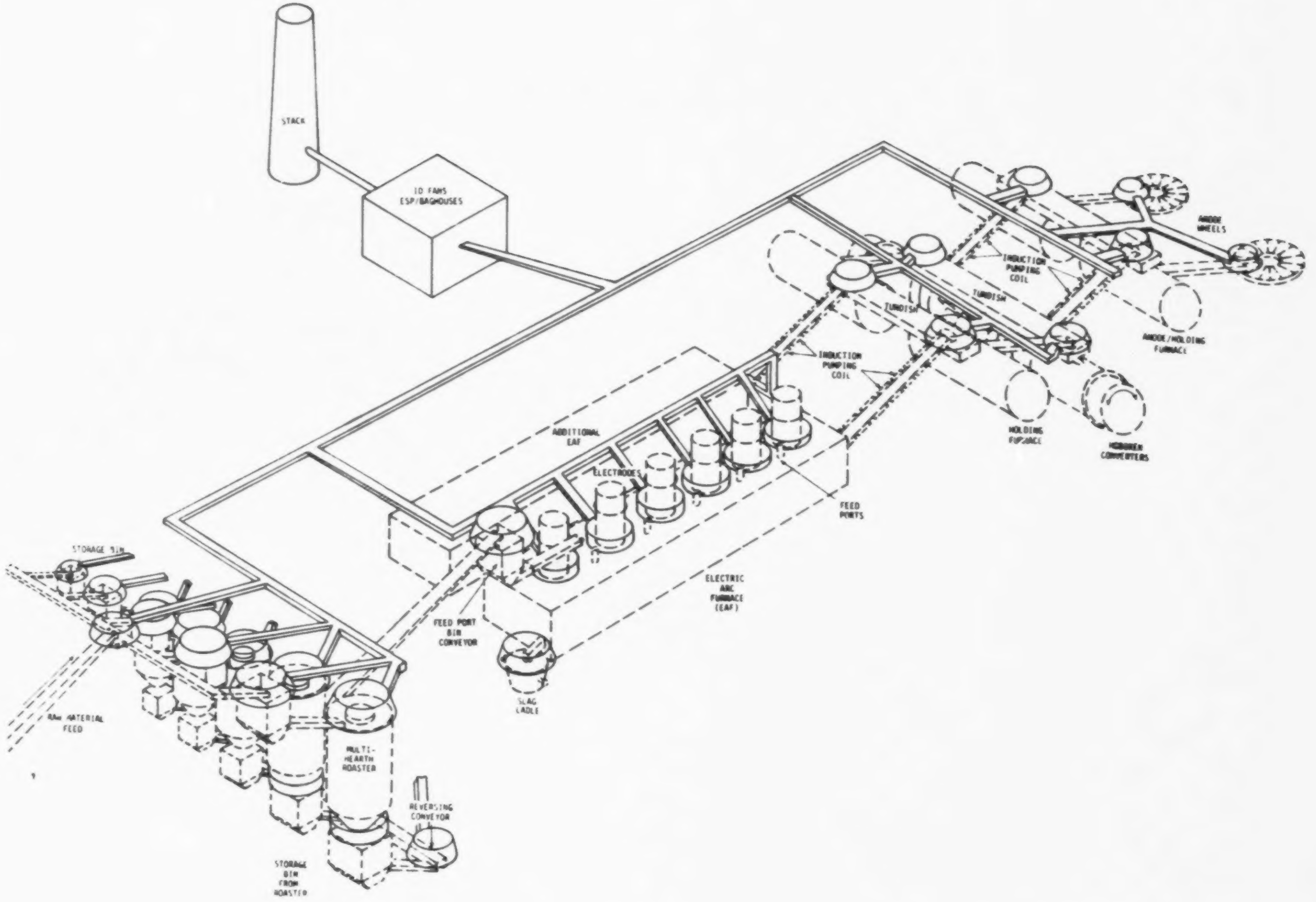

STACK
ID FANS
ESP/BAGHOUSES
ANODE
WHEELS
INDUCTION
PUMPING
COIL
TUNDISH
TUNDISH
ANODE/HOLDING
FURNACE
INDUCTION
PUMPING
COIL
HOLDING
FURNACE
HOBOKEN
CONVERTERS
ADDITIONAL
EAF
ELECTRODES
FEED
PORTS
STORAGE BIN
ELECTRIC
ARC
FURNACE
(EAF)
FEED PORT
BIN
CONVEYOR
SLAG
LADLE
RAW MATERIAL
FEED
MULTI-
HEARTH
ROASTER
REVERSING
CONVEYOR
STORAGE
BIN
FROM
ROASTER

Figure 21b. Fugitive emission collection system for cascading/induction/gravity flow.

RELATIVE COSTS, EMISSIONS, AND ENERGY EVALUATION
EXISTING VERSUS ALTERNATE SYSTEMS

Cascading/Induction/Gravity System Versus Conventional Primary Copper Smelter

Cost

Building costs would be lower.
Operational costs would be higher.

Emissions

Minimal

Energy Requirements

Minimal for gravity transportation from furnace to furnace
High than crane transportation during induction pumping

Pros and Cons

Maintenance would be higher.
Proximity of furnaces to each other
Change in operating techniques

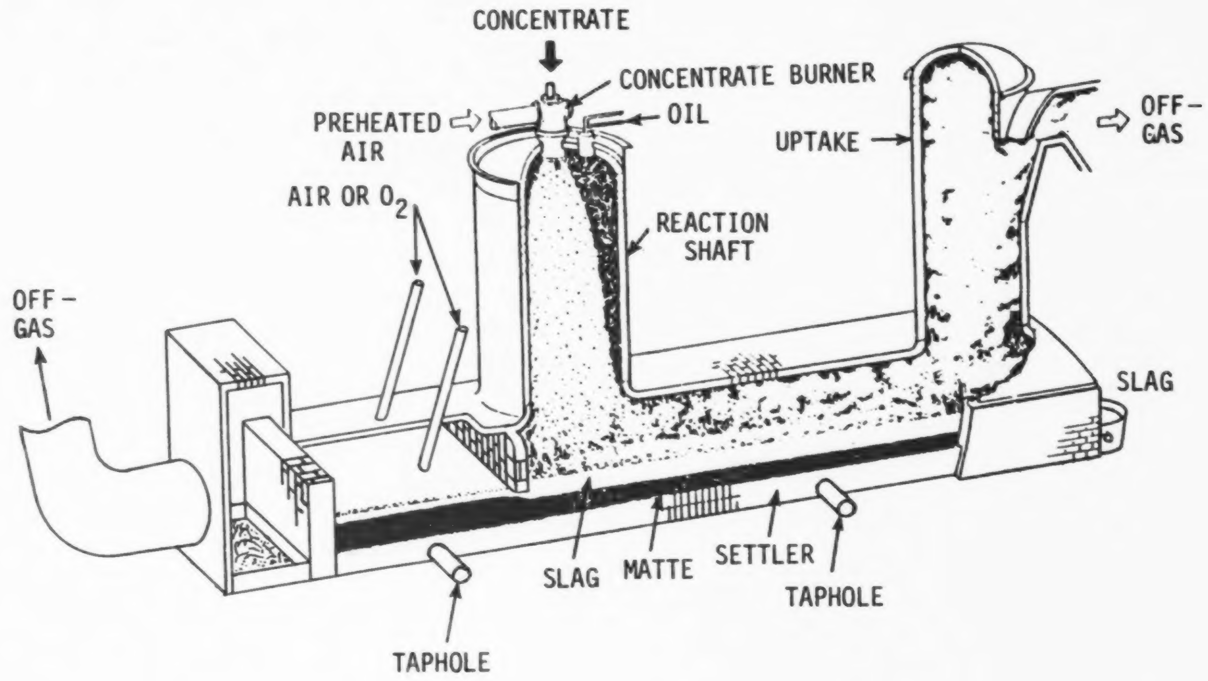

CONCENTRATE
CONCENTRATE BURNER
PREHEATED AIR
OIL
UPTAKE
OFF-GAS
AIR OR O_2
REACTION SHAFT
OFF-GAS
SLAG
SLAG
MATTE
SETTLER
TAPHOLE
TAPHOLE

Figure 22. Modified Outokumpu.

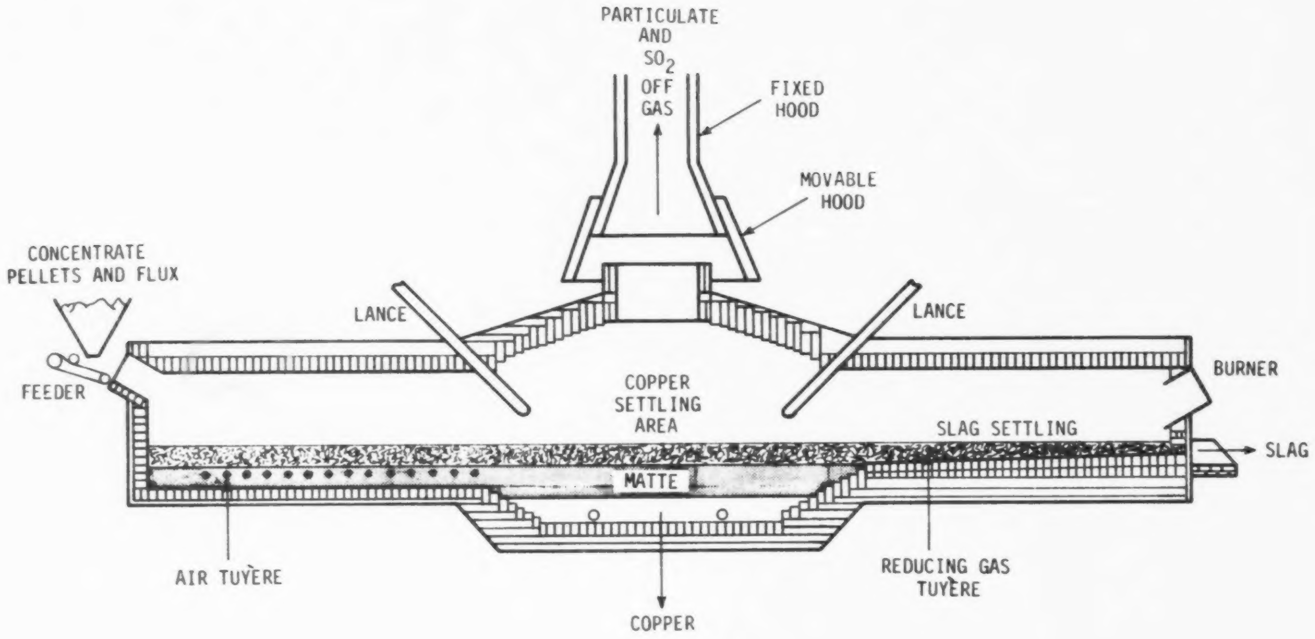

PARTICULATE
AND
SO_2
OFF
GAS
FIXED
HOOD
MOVABLE
HOOD
CONCENTRATE
PELLETS AND FLUX
LANCE
LANCE
FEEDER
COPPER
SETTLING
AREA
BURNER
SLAG SETTLING
SLAG
MATTE
AIR TUYÈRE
REDUCING GAS
TUYÈRE
COPPER

Figure 23. Modified Noranda.

RELATIVE COSTS, EMISSIONS, AND ENERGY EVALUATION
EXISTING VERSUS ALTERNATE SYSTEMS

Modified Outokumpu or Noranda Versus Smelting and Converters

Costs

Building costs would be less.
Operational costs would be less.

Emissions

Less

Energy Requirements

Less

Pros and Cons

New operating techniques
Refractory problems
Loss of one unit is loss of total production.

RELATIVE COSTS, EMISSIONS, AND ENERGY EVALUATION
EXISTING VERSUS ALTERNATE SYSTEMS

Flash Smelters, Noranda, and Electric Arc Furnaces Versus Reverberatory Furnaces

Costs

Building costs could be greater.
Operational costs slightly higher.

Emissions

Less

Energy Requirements

More efficient smelting

Pros and Cons

Change in operating techniques
Maintenance could be greater
Better control

ENCLOSURES FOR INDIVIDUAL FURNACES

Each individual furnace (Figure 24) would be enclosed and self-contained with its own captive EOT crane and a transfer car (for ladle additions, etc.), tuyere puncher, and fugitive emissions system. The equipment would all be remotely controlled except during maintenance, when it could be effectively controlled manually. If needed, closed circuit televisions would permit the operator to monitor all operations. As envisioned, the operator would work from a control booth or room outside of the enclosure. For example, the operation of a converter might be as follows: a covered ladle of matte from the smelter furnace (reverberatory, electric arc, etc.) would be sitting on a transfer car outside of the smelting furnace enclosure, having been picked up by an EOT crane and positioned on the transfer car outside of the enclosure of the converter. The operator would raise the sliding door and move the transfer ladle car inside the converter enclosure and the fugitive emission evacuation would be energized. The sliding door would then be closed and the ladle cover removed with a jib boom-type hoist. The ladle would then be engaged by the hooks on the crane, lifted, and poured into the converter. Then the ladle would be returned to the transfer car and the procedure reversed until a sufficient number of ladles of matte, reverts, etc., are put into the converter. Blowing would then begin. The same operating methods that are now employed would then be applied. It is true that the operator would now be working outside the enclosure and might feel he does not have good control of the operation, but he would shortly become acclimated. Control of fugitive emissions from a small confined area would be easier than from the entire building, where open doors, etc., cause disruptions in the flow of the fugitives to the overhead monitor collection system. The energy expenditure for air movements also would be lower, whereas building costs would be a little higher.

CRANE EVACUATION OF FUGITIVE EMISSIONS

As shown in Figure 19 this might be a relatively easier way of capturing fugitive emissions than the use of an enclosed building evacuation. The principle entails the use of a spreader beam with a telescopic emission capture duct built as portion of the beam. The telescopic duct would act in the same manner as the stiff-leg telescopic soaking pit crane used in the steel industry. The telescopic duct would be attached to a fixed duct (under the trolley), which in turn is ducted to another transfer duct attached to one of the crane girders. This duct would be rectangular, with a split rubber cover that allows the nozzle from the proceeding duct to discharge into it with a minimal opening. The transfer duct on the crane girder would in turn discharge to another duct, either on the crane runway, walkway, or overhead to the building trusses. This duct would discharge at a number of points to a fixed collection system. The discharge fan could either be fixed on the trolley, the crane bridge girder, or the building. Another possibility would be to discharge the fugitives via a duct directly above the trolley and to use a central monitor at the peak of the building trusses to collect the fugitive emissions and discharge them to the main collection system.

The telescopic fugitive emission capture system would be in operation during the picking up of a ladle filled with matte, blister copper, or con-

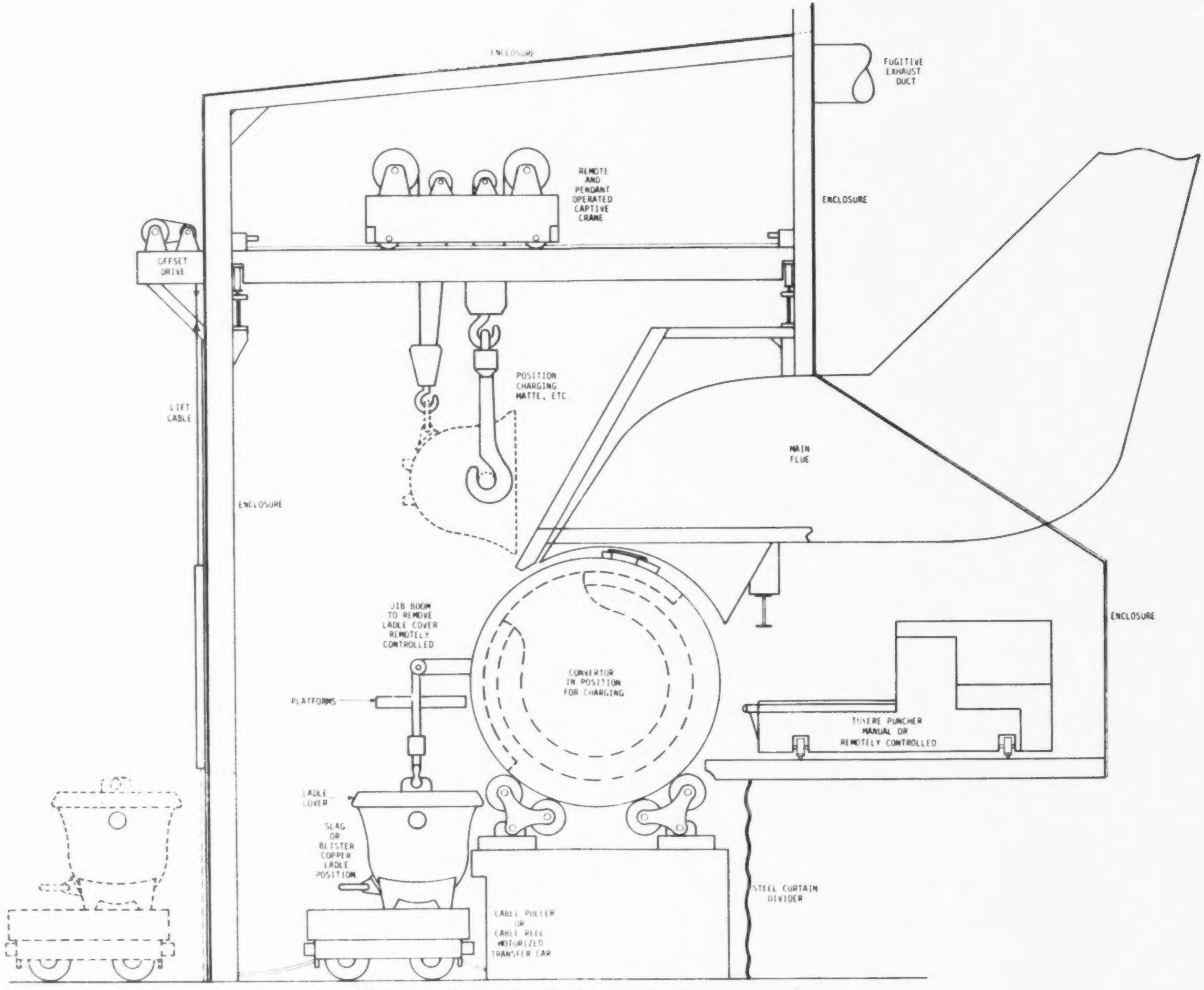

ENCLOSURE
FUGITIVE EXHAUST DUCT
REMOTE AND PENDANT OPERATED CAPTIVE CRANE
ENCLOSURE
OFFSET DRIVE
POSITION CHARGING MATTE, ETC.
LIFT CABLE
MAIN FLUE
ENCLOSURE
JIB BOOM TO REMOVE LADLE COVER REMOTELY CONTROLLED
ENCLOSURE
CONVERTOR IN POSITION FOR CHARGING
PLATFORMS
TUYERE PUNCHER MANUAL OR REMOTELY CONTROLLED
LADLE COVER
SLAG OR BLISTER COPPER LADLE POSITION
STEEL CURTAIN DIVIDER
CABLE PULLER OR CABLE REEL MOTORIZED TRANSFER CAR

Figure 24. Individual furnance enclosure.

RELATIVE COSTS, EMISSIONS, AND ENERGY EVALUATION
EXISTING VERSUS ALTERNATE SYSTEMS

Enclosing Each Furnace Versus Existing Open Layout

Costs

Building costs may be higher.
Operational costs may be higher.

Emissions

Minimal

Energy Requirements

Slightly greater

Pros and Cons

Preventive maintenance must be practiced.
Air movements or changes must be properly controlled.
Change in operating techniques.

Enclosing Each Furnace Versus Enclosed Building Evacuation

Costs

Building and evacuation equipment costs would be less.
Operational costs would be less.

Emissions

In each case minimal

Energy Requirements

Would be less

Pros and Cons

Control of emissions at source.
Preventive maintenance must be practiced.
Air movements or changes not as critical.
Change in operating techniques.

RELATIVE COSTS, EMISSIONS, AND ENERGY EVALUATION
EXISTING VERSUS ALTERNATIVE SYSTEMS

Ladle Emission Control Using Covered Bottom Pour Ladles and Crane Evacuation Versus Secondary Hooding, Building Evacuation, etc.

Costs

Building costs should be less.
Operational costs should be less.

Emissions

Less than secondary hooding
Same as building evacuation

Energy Requirements

Less

Pros and Cons

Special crane and duct design
Special ladle design
Higher maintenance
Greater handling of ladles
Change in operational techniques

verter slag; the transporting of ladles between process furnaces; the pouring of the matte, blister copper, or converter slag into the respective vessels; and also during the time the ladle is being filled. With a good capture velocity, designed controlled leaks, and good design, this technique would minimize fugitive emissions during ladle handling.

HOBOKEN CONVERTERS (FIGURE 18) WITH SWINGAWAY HOODS

As shown in Figure 18, the use of Hoboken converters with swingaway hoods would minimize fugitive emissions over the Peirce-Smith converters during slagging, pouring, blowing, etc. The Hoboken converter with a properly designed draft control and gas take-off should be fugitive emission free during blowing operations. During slagging and blister copper pouring, however, the use of a swingaway fugitive emission hood would minimize secondary emissions. Emission problems during matte charging could be handled by a telescopic capture duct attached to the spreader beam of the EOT crane. A covered ladle in conjunction with a bottom-pour stopper rod also could be used to minimize fugitive emissions during transporting and discharging the molten metal or slag to a respective furnace. The ladle cover would be placed on at the station where the ladle is filled. The cover would be installed under some sort of emission collection hood, either fixed or movable. The base of the ladle would be adapted to allow the ladle to be discharged into the receiving vessel, and if the receiving vessels were operated at a slightly negative draft, emissions would be minimal. The operation of the stopper rod or the ladle would be handled by the crane operator via an "auto-pour unit," which is used for teeming operations in the steel industry (Figure 25).

ENCLOSED BUILDING EVACUATION TO A CENTRAL DISCHARGE SYSTEM

Enclosed building evacuation, with provisions to provide air changes as required in the various areas of the building (for the operation and maintenance personnel) as well as air for the process combustion and in-leakage, is another possible way to minimize fugitive emissions. (See Appendix Table B-2 for a summary on enclosed building evacuation.)

OTHER POSSIBLE EMISSION CONTROL SYSTEMS

- Floor operated charging machines (Figure 26a, 26b) or manipulators installed under hooded enclosures in the vicinity of the process furnaces
- Q-BOP furnaces (Figure 27) (as used in the steel industry), using a central injection lance for oxygen-enriched air and feed, installed in a hooded enclosure with primary and secondary hoods
- INCO flash smelting furnace (Figure 28) (would be essentially the same category as the Noranda smelter)

Table 6 shows the author's estimates of energy requirements for the use of secondary type hoods, individual furnace enclosure, or building evacuation. The numbers indicate energy usage per unit; thus, if two reverberatory

RELATIVE COSTS, EMISSIONS, AND ENERGY EVALUATION
EXISTING VERSUS ALTERNATE SYSTEMS

Hoboken Converters Versus Peirce-Smith Converters (Equivalent Tonnage)

Costs

Building costs would be approximately the same.
Operational costs could be less.

Emissions

Much less

Energy Requirements

Less

Pros and Cons

Maintain refractory
Change in operating technique
Higher operating maintenance

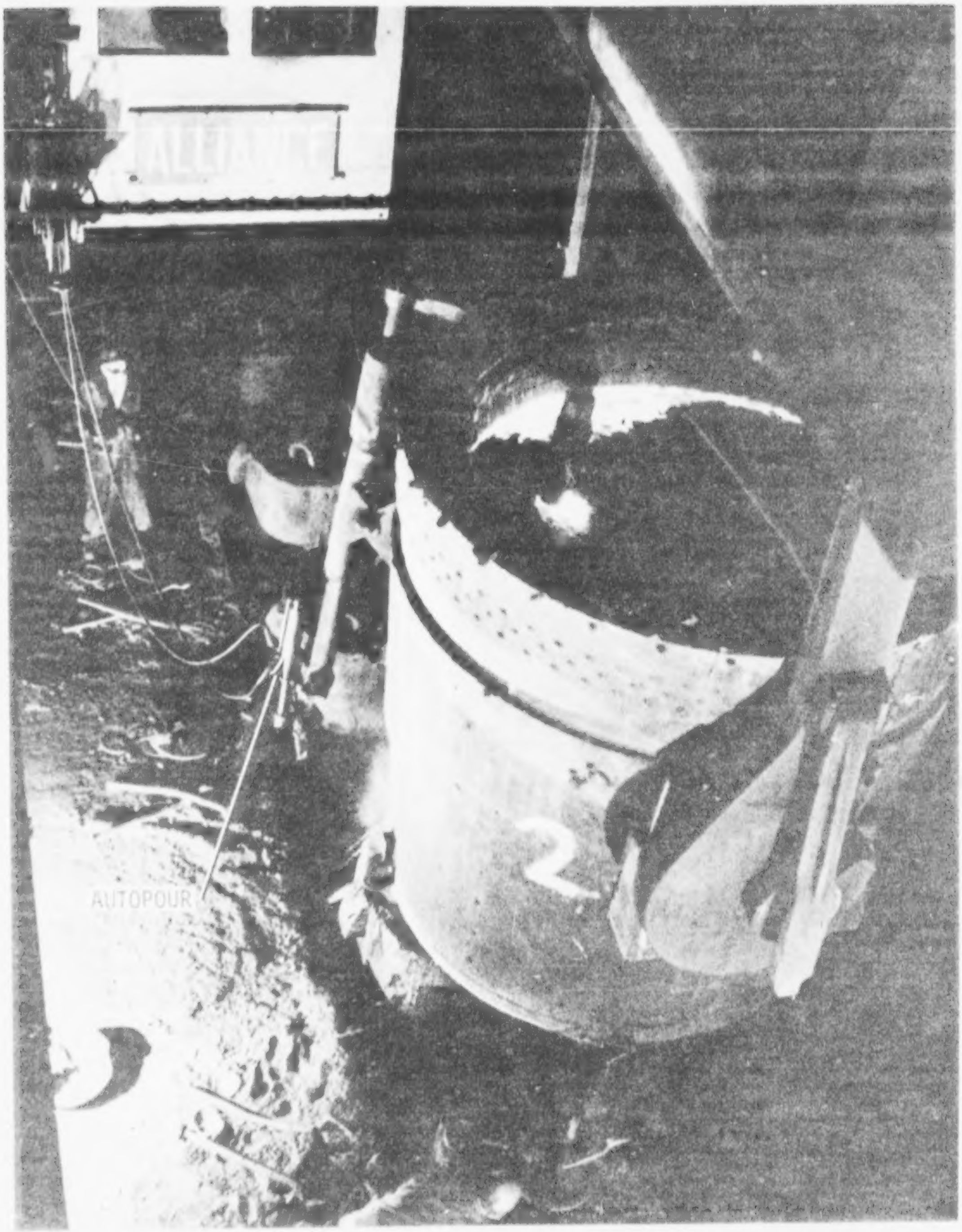


Figure 25. Hydraulic cylinder mounted on barrel of ladle rigging raises and lowers stopper rod to control flow of molten steel from ladle to ingot mold. Source: "Making, Shaping and Treating of Steel" by U.S. Steel, Edition 1971. Copyright by U.S. Steel Corp. 1971. Reproduced with permission.

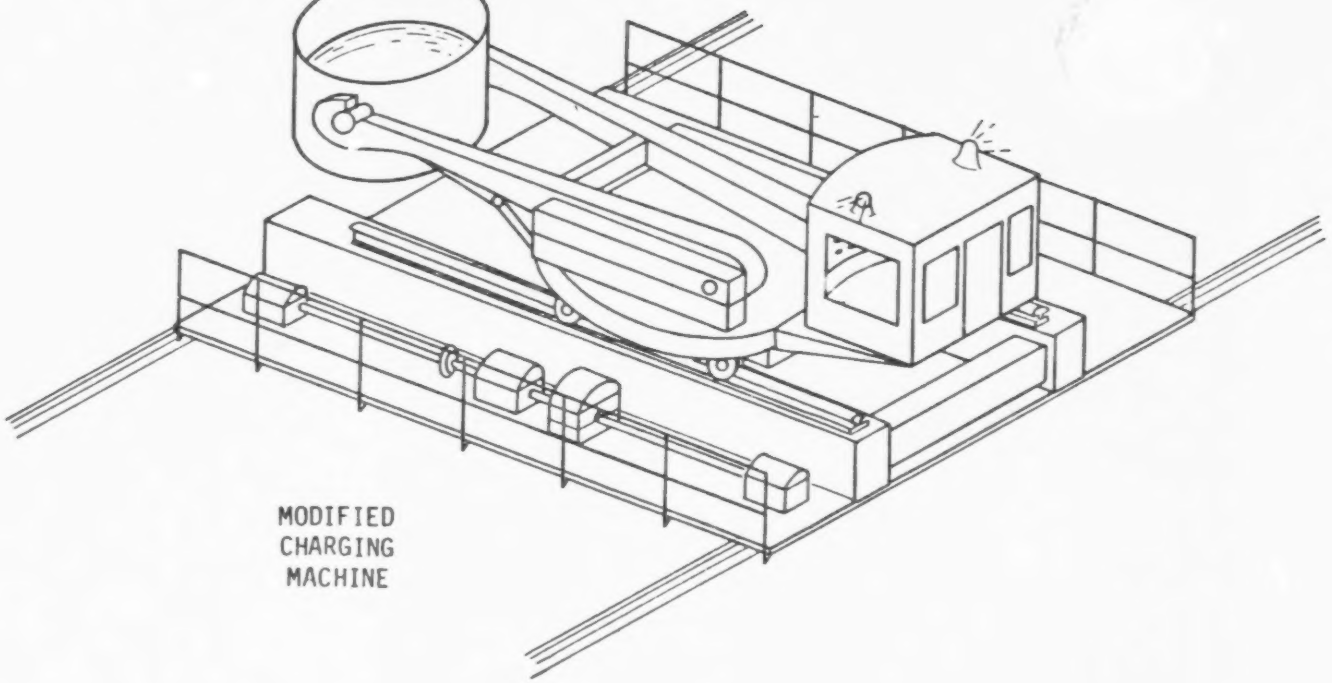
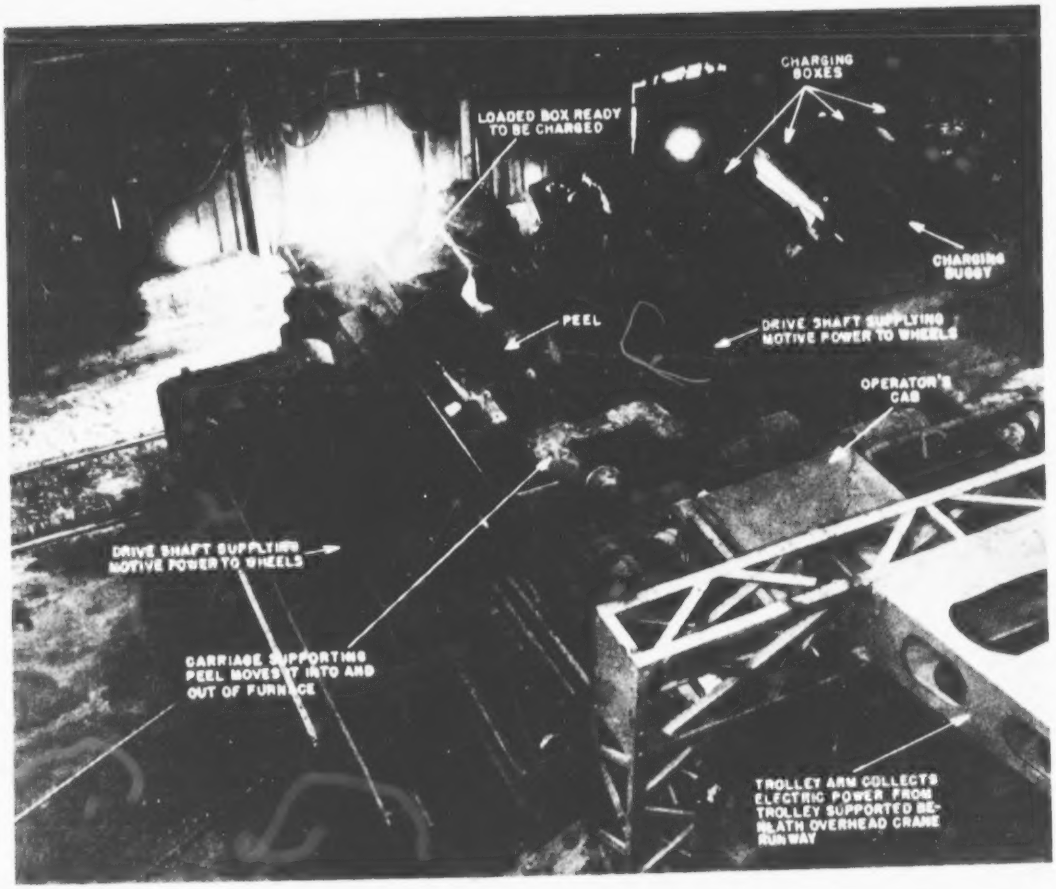

MODIFIED
CHARGING
MACHINE

Figure 26a. Modified charging machine.



CHARGING BOXES
LOADED BOX READY TO BE CHARGED
CHARGING BUGGY
PEEL
DRIVE SHAFT SUPPLYING MOTIVE POWER TO WHEELS
OPERATOR'S CAB
DRIVE SHAFT SUPPLYING MOTIVE POWER TO WHEELS
CARRIAGE SUPPORTING PEEL MOVES IT INTO AND OUT OF FURNACE
TROLLEY ARM COLLECTS ELECTRIC POWER FROM TROLLEY SUPPORTED BENEATH OVERHEAD CRANE RUNWAY

Figure 26b. Typical charging machine in the steel industry. Source: "Making, Shaping, and Treating of Steel," by U.S. Steel, 1971. Copyright by U.S. Steel Corp. 1971. Reproduced with permission.

RELATIVE COSTS, EMISSIONS, AND ENERGY EVALUATION
EXISTING VERSUS TOTAL BUILDING EVACUATION

Costs

Building costs would be greater.
Operational costs would be greater.

Emissions

Less

Energy Requirements

Greater

Pros and Cons

Air changes and associated problems
Maintenance would be greater.
Operational changes

RELATIVE COSTS, EMISSIONS, AND ENERGY EVALUATION
EXISTING VERSUS ALTERNATE SYSTEM

Floor-Operated Chargers With Furnace Emission Hooding Versus Existing System of Ladle Handling Transporting, Charging, etc.

Costs

Building costs would be higher.
Operational costs would be higher.

Emissions

Less

Energy Requirements

Greater because of emission hooding

Pros and Cons

New operational techniques
Maintenance higher

RELATIVE COSTS, EMISSIONS, AND ENERGY EVALUATION
EXISTING VERSUS ALTERNATE SYSTEMS

Q-BOP Versus Smelters and Converters

Costs

Building costs would be less.
Operational costs should be less.

Emissions

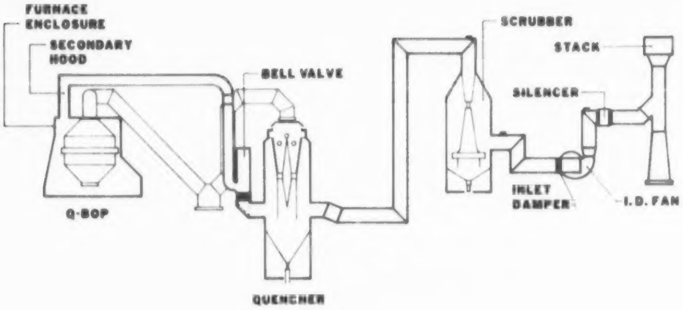
Less

Energy Requirements

Similar

Pros and Cons

New operating techniques
Higher maintenance
Higher tonnages
Good enclosures required


FURNACE ENCLOSURE
SECONDARY HOOD
BELL VALVE
Q-BOP
QUENCHER
SCRUBBER
STACK
SILENCER
INLET DAMPER
I.D. FAN

Gas cleaning system

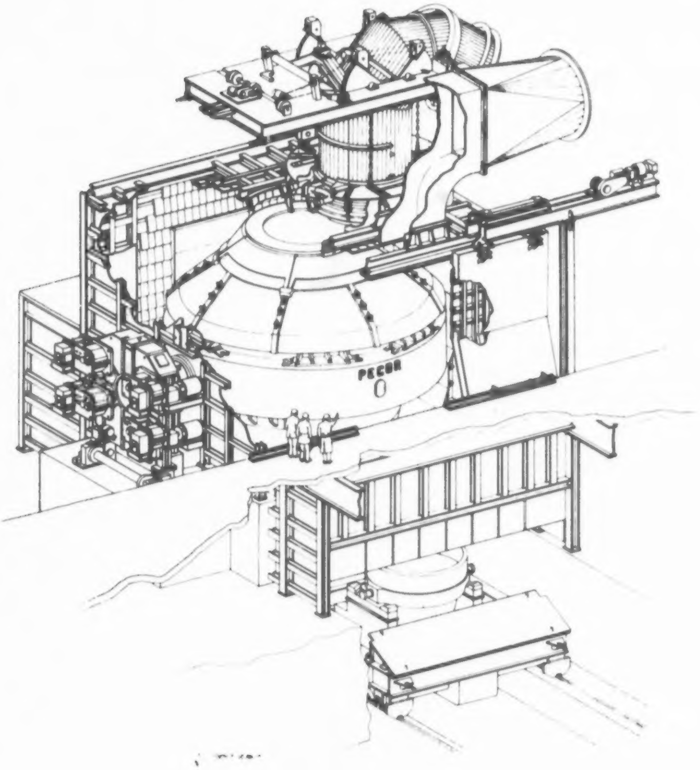


Figure 27. The 225-ton Q-BOP is enclosed in a doghouse to prevent fugitive emissions. Source: Iron and Steel Engineer, November 1978, reprinted with permission.

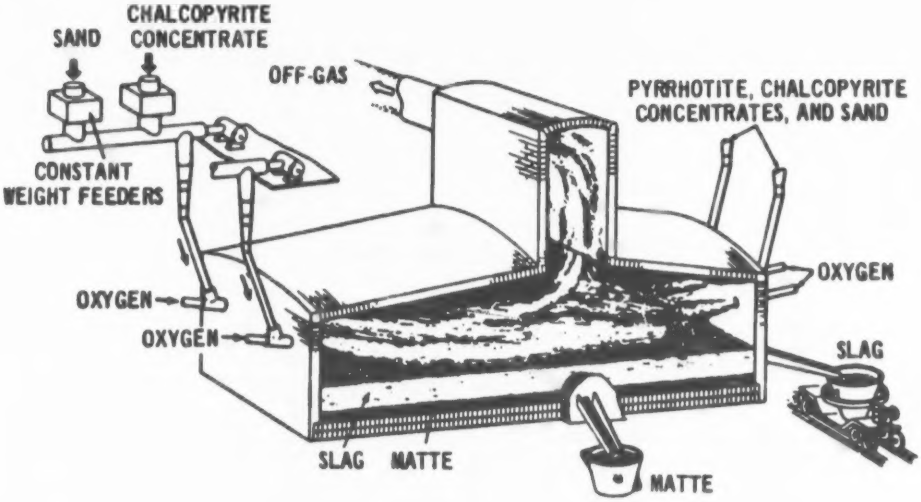

SAND
CHALCOPYRITE CONCENTRATE
OFF-GAS
PYRRHOTITE, CHALCOPYRITE CONCENTRATES, AND SAND
CONSTANT WEIGHT FEEDERS
OXYGEN
OXYGEN
OXYGEN
SLAG
SLAG
MATTE
MATTE

Figure 28. INCO flash smelting furnace.

TABLE 6. ENERGY REQUIREMENTS FOR CONTROL OF FUGITIVE (GASEOUS AND PARTICULATE) EMISSIONS AT INTAKES AND DISCHARGE POINTS

Process equipment	kWh/h (minimum[a]/maximum[b])		
	Secondary hoods; capture hoods	Individual[b,c] enclosure/unit	Building[b,c] evacuation
Roaster	192/200	27	271
Dryer	156/162	38	38
Outokumpu	30/149	90	411
Noranda	60/149	50	172
Electric arc furnace	89/176	110	444
Reverberatory	53/149	90	355
Peirce-Smith	168/216	11	365
Hoboken	17/108	17	305
Anode	55/108	11	365
EOT crane	7/73	-	-

[a] Operation only when emissions are being captured.

[b] Continuous operations.

[c] Air changes estimated at 12 per hour.

Note: Amounts shown are per unit and with a baghouse.

furnaces were required, the secondary and individual energy would double, whereas building evacuation would be 75 to 80 percent greater.

Appendix A shows the costs attributed to secondary hooding systems for converters. In Appendix B, Table B-1 shows the estimated hooding efficiencies, and Table B-2 presents a summary of current fugitive emission control systems.

APPENDIX A

COSTS OF SECONDARY HOODING SYSTEMS FOR CONVERTERS

This section includes a capsule discussion on costs of the following major types of secondary hoods:

- Fixed type -- made of structural steel with an elliptical cross-section. It is attached to the primary or uptake hood.
- Fixed and movable -- consist of a movable intermediate hood and a hood fastened to the gate. Both hoods are made of structural steel with elliptical cross-sections so that they telescope in the retracted mode.
- Swingaway type with a fixed overhead hood -- made of structural steel, refractory-lined, and supported by pivot arms with a motorized drive to permit positioning during blowing and pouring operations.
- Combination of fixed, movable, and swingaway types.

Costs of Fugitive Emission Control Systems

Cost parameters - secondary hooding --

This cost section includes various items that must be installed or modified to achieve control of fugitive emissions from the Peirce-Smith converter (this furnace being the worst source of secondary emissions). It does not include costs of certain operating procedures that would minimize fugitive emissions, e.g., allowing minimal clearance between the primary uptake hood and the apron of the converter, or maintaining proper matte charges to provide for direct flow of gases from the mouth of the converter to the centerline of the primary uptake hood.

The following items are considered in this cost section:

- A fixed hood having an elliptical cross-section of 17 feet 6 inches on the major axis and 7 feet on the minor axis, and 9 feet 6 inches long. The plate is 3/8-inch carbon steel, with stiffeners of 7-inch channels. The fixed hood is bolted to the primary hood and to the smoke plenum of the secondary duct system.
- A movable hood in the retracted position, parked above the fixed hood. It has an independent track system with a five-speed, double-grooved hoist unit and slack cable limit switch. The movable hood

Finish - Scour and heat set.

Nomex cloth is used in the slag fuming baghouse due to its ability to withstand temperatures up to 230°C. The normal operating gas temperatures are about 200°C with odd peaks near the maximum. Apart from temperature considerations, these bags are subjected to more frequent mechanical shaking than the other baghouses and a bag life of just over 2 years is normal. This shorter life combined with a cloth cost four times that of terylene raises doubts on the economics of operating baghouses at higher temperatures instead of increasing the gas cooling facilities. The specifications of the nomex cloth used are as follows -

Weave - 2 x 2 twill
Weight - 300 grams per square metre
Permeability - 16 cubic metres per square metre per minute at 25 millimetres water gauge pressure.

Shaking Arrangement - Dust build up on the cloth in our process gas baghouses is removed by means of mechanical shaking applied to the top section of the bags. Various types of shaking arrangements have been tried over the years, but we have standardised on hanging the bags to a moveable top frame which is given a sideways rocking movement through an electrically driven crank disc. The length of stroke is fixed at 178 mm and the motor speed is 92 revolutions per minute. Shaking time may be varied but it is kept to a minimum to reduce bag wear and is presently controlled at 10 seconds shake every 4 hours. The chamber being shaken is isolated for 15 seconds before the shake commences and 120 seconds after the shake cycle to allow dislodged dust to settle into the screw conveyor troughs.

The new centralised baghouse will be on automatic control for shaking and it will be possible to vary the cycles as required. Some of the chambers in the existing blast furnace baghouse have been converted to automatic control and a typical chart indicating gas temperature, gas volume and pressure drop across the chamber is shown in Figure 3. The step trace in the pressure drop and gas volume patterns reflects the changes as other chambers are shaken in sequence. It is hoped that a much better control can be exercised over our baghouse operations by having these recording facilities.

Small Prefabricated Baghouses - Several smaller type packaged baghouses are in use on the Works and we are generally satisfied with their operation. Our first baghouse using terylene needlefelt cloth and employing a reverse air pulse system of bag cleaning was installed in 1975, to collect an arsenic-bearing fume arising from our refining operations. It has performed to expectations and confirmed that the higher filter velocity of 12 millimetres per second is satisfactory for this particular fume. Attention was given to designing a bag-changing arrangement whereby the operators would not be over-exposed to the high arsenic fume collected. A vacuum cleaning system was installed which cleans the outside of the bag as it is withdrawn from the top of the chamber. The system has worked satisfactorily under close supervision. Bags are changed as holed and the failures to date have been caused by burning cinders getting into the baghouse in the gas stream and some rubbing of the cloth on the support frames. It is estimated that a normal bag life of 4 years should be obtained.

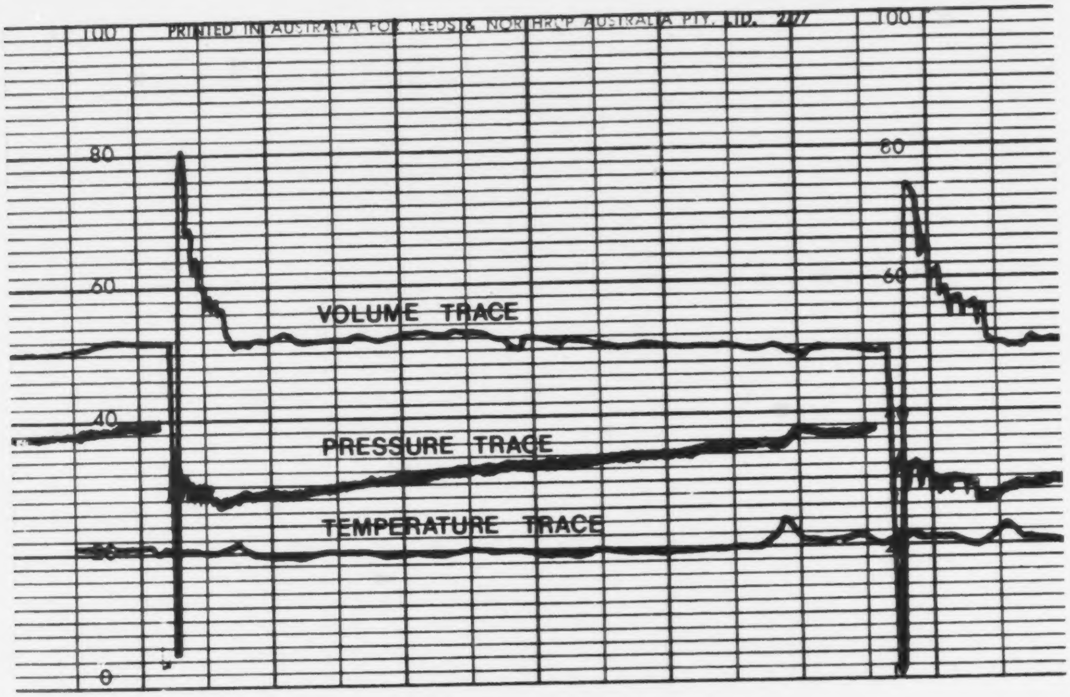


Figure 3 — Typical Control Chart

A similar baghouse using the reverse air pulse cleaning method was installed to handle the dust and fume arising from the zinc melting furnace. The fume originates from the use of ammonium chloride flux in the furnaces and in combination with acidic electrolyte sticking to the cathode zinc sheet, presented some filtering problems. Extra heat was bled into the system from an adjacent combustion furnace to keep the gas temperature above the dew point. The baghouse is performing satisfactorily and apart from some problems with bags rubbing on the wire cages, it is expected that a bag life of about 3 years will be achieved. Present cloth specifications are as follows -

Type - heavy duty terylene needlefelt
Weight - 400 grams per square metre
Permeability - 16 cubic metres per square metre per minute at 25 millimetres water gauge pressure.

General Operating Comments - Men working inside baghouses on bag maintenance are highly exposed to particulates and with lead fume the health risk is correspondingly greater. They have to be provided with a full cover of protective equipment including suitable respirators and supervision should be strong. With the new centralised baghouse it is hoped that all bag maintenance will be done on daywork with one shiftworker controlling shift operations. The total number of men employed in baghouse work will be reduced by at least 4 and perhaps 6 men.

The problem in continuously monitoring particulate material in the filtered gases has not yet been resolved as preliminary trials with an optical instrument were unsuccessful and tests are now being conducted with an IKOR monitor. This instrument measures the charge transfer which occurs when particles collide with a sensor in the gas stream. Normal emissions will be in the order of 10 milligrams particulates in each cubic metre of effluent gas and this is a very low level for detection. An instrument capable of indicating changes in emission levels is required to enable bag maintenance personnel to locate leaking baghouse chambers.

Wet Scrubbers

Wet scrubbing has been practised in the lead smelter for many years and has mainly been used in the sinter plant area to handle the wet hygiene gases. The particulates contained in these gases are of a sticky nature and caused blockage problems both in the ducting and in the scrubber. While the units were clean and working properly the emissions were reasonable, but the position soon deteriorated and frequent plant stops were necessary to control the problem. At this time a modified scrubber incorporating the jet impingement and fluidised bed principles was adopted. The Scrubber design is shown in Figure 4 and illustrates how the entering gas stream impinges on the water surface before it rises and passes through two beds of marbles mounted on retaining grids. Water sprays are situated on the underside of the marbles and also at the top of the scrubber to wash away the adhering particulates. Unfortunately the marble beds tended to block up with mud and the cleaning operation during plant stoppages was very messy. Venturi scrubbers have now replaced the marble scrubbers and plugging is avoided by the relatively open design of these units. Another type of scrubber tried unsuccessfully in the sinter plant and subsequently installed in the leaching section of the zinc plant is the Joy Microdyne scrubber. A diagramatic sketch outlining its main components is shown in Figure 5. This low energy scrubber is unable to achieve the collection efficiency required in handling the hot fume laden gases from the drying of leach residue material and will ultimately be replaced by a venturi scrubber.

Venturi Scrubbers - High energy scrubbers were first installed in the zinc plant to handle the gas from the zinc oxide roasting kilns. A unit consists of two venturis in series which are followed by a cyclone separator and the gas is then discharged through a fan up the stack. The first venturi is designed to cool and condition the gas while the major scrubbing action is achieved in the second venturi. A volume of 240 cubic metres per minute is handled by the fan and water additions are 70 litres per minutes in the first stage and 150 litres in the second stage. Pressure drops are approximately 1120 millimetres water gauge across the venturis and 150 millimetres water gauge across the cyclone separator. The fan is driven by a 130 kilowatt electric motor. Unfortunately, the collection efficiency of these units is slightly below 99 percent and the particulate emissions do not meet the Clean Air Regulation limits. It is now intended to replace one of the existing units with a long throat venturi to determine if better results are possible with the longer throat design.

A long throat venturi scrubber of the Stairmand - I.C.I. design[1] has been used successfully in the sinter plant hygiene circuit and was recommended by our Consultant, Charles G. Martin[2]. The design features are shown in Figure 6 and the unique characteristic is that the throat length should be at least seven throat diameters. This is based on the theory that the turbulence created in the longer throat section will ensure intimate contact of the water and dust particles. Under these conditions the method of water injection is not considered important and simple tangential entry is adequate.

Initial design parameters for the sinter plant units were established from a test unit set up in the plant. The specifications for the final arrangement was then submitted by our consultant and six similar units were installed. They were designed to handle a nominal 250 cubic metres per minute of air, throat diameter to be 240 millimetres and a pressure drop across the installation of 660 millimetres. Water requirement was estimated to be 154 litres per minute and a 50 kilowatt power requirement to drive the fans was calculated.

In practice the volume handled is around 180 cubic metres per minute but the other specifications are substantially correct. Collection efficiencies of 99.8 percent by weight are common and blockages have been eliminated with consequent better draughting. However, there are still problems to be solved in the fan operation as there is a carryover of dirty water droplets from the cyclone separator and the solids build up on the fan blades causes out of balance problems. The fan blades also show the effects of corrosion and erosion, and alternative materials of construction are being tried. It is possible that a mist eliminator will be required on top of the cyclone separator. We still have some way to go before mastering the techniques of venturi scrubber operation.

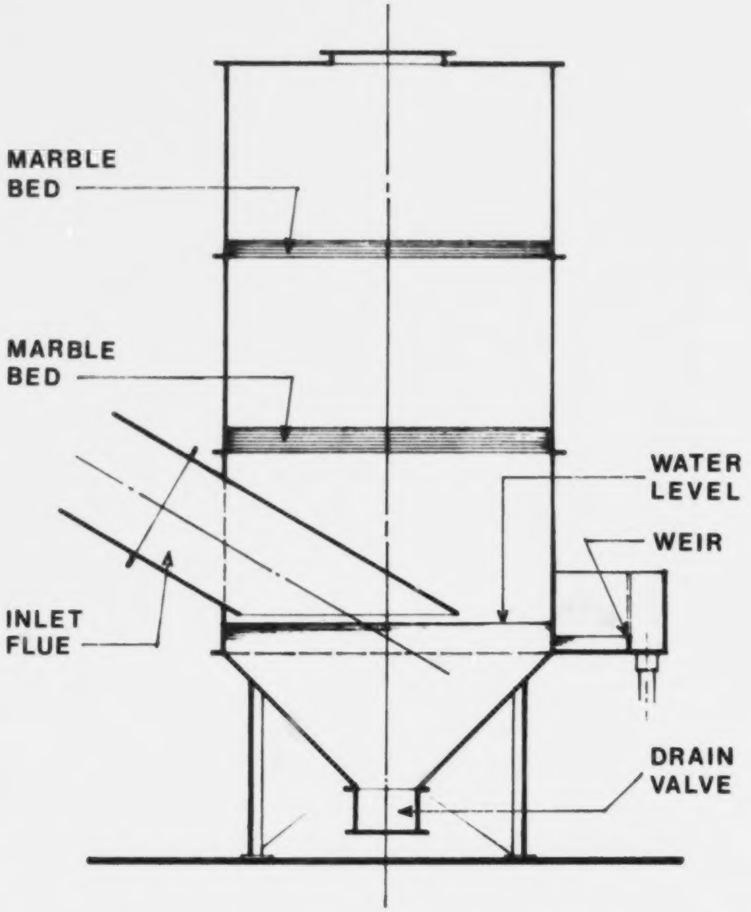

MARBLE BED
MARBLE BED
WATER LEVEL
WEIR
INLET FLUE
DRAIN VALVE

Figure 4 — Marble Type Scrubber

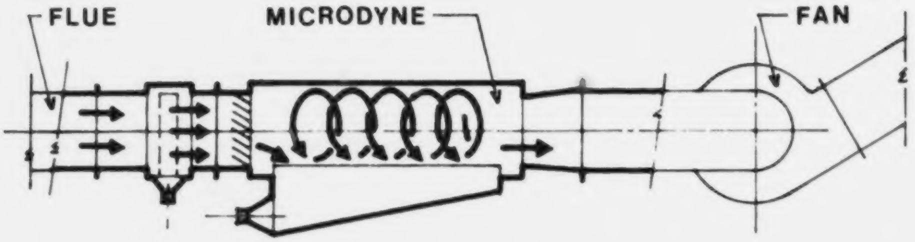

FLUE
MICRODYNE
FAN

Figure 5 — Joy Microdyne Scrubber

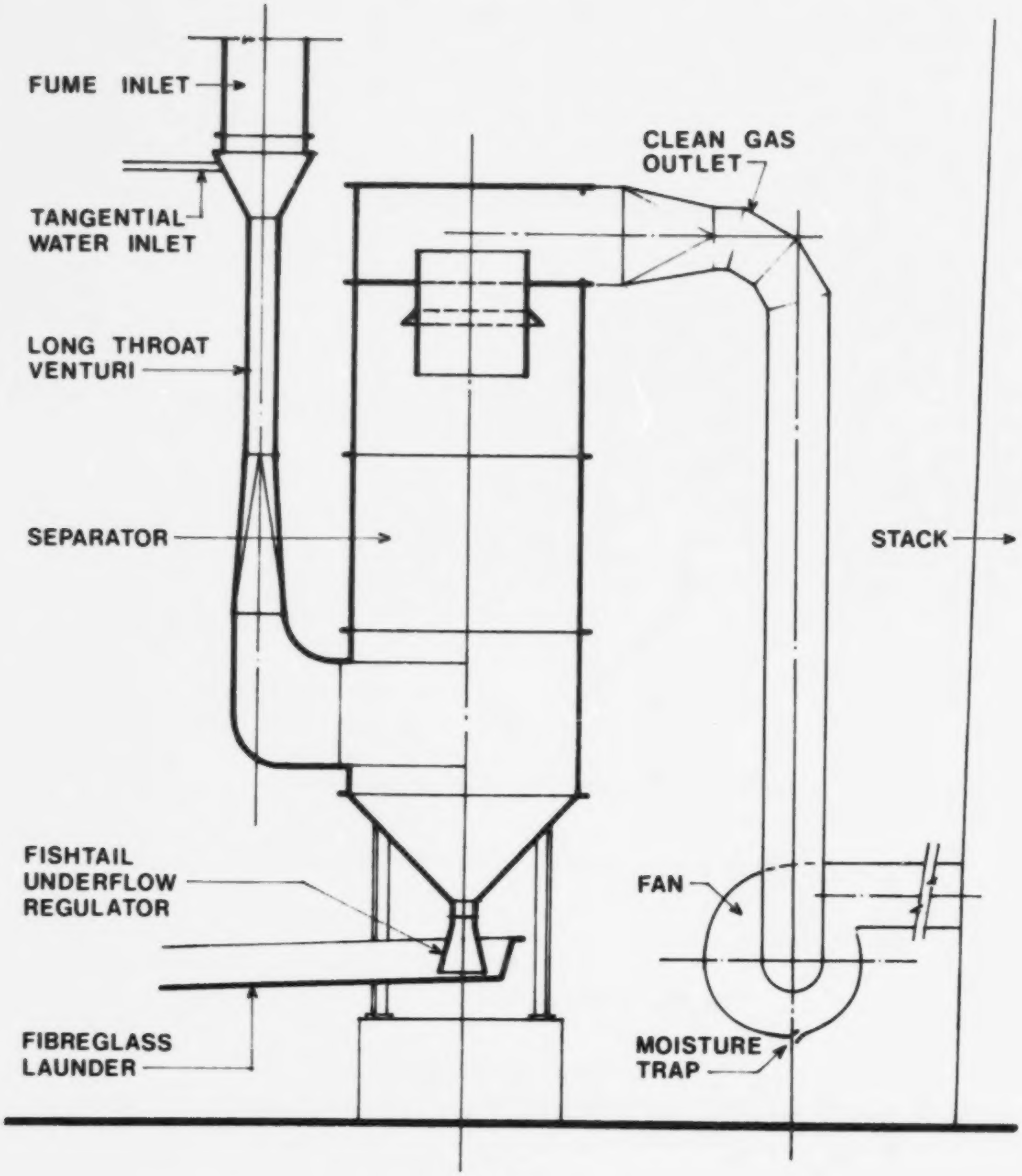

FUME INLET
CLEAN GAS OUTLET
TANGENTIAL WATER INLET
LONG THROAT VENTURI
SEPARATOR
STACK
FISHTAIL UNDERFLOW REGULATOR
FAN
FIBREGLASS LAUNDER
MOISTURE TRAP

Figure 6 — Long Throat Venturi Scrubber

CONCLUSIONS

B.H.A.S. is committed to heavy expenditure on environmental projects at the present time and there is still much to be done. The toxicity of lead is very much under scrutiny and emission standards are being progressively reduced, making it difficult for older type plants to comply. Serious consideration must now be given to the new technology required in the future to meet changed environmental considerations, otherwise control costs will become prohibitive.

The Management of B.H.A.S. is very aware of its obligations and is maintaining a policy designed to control particulate emissions to the limits of the available equipment.

ACKNOWLEDGEMENTS

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APPLICATIONS OF BAGHOUSES IN LEAD AND ZINC SMELTERS

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ABSTRACT

Fabric filters are widely and successfully used in lead-zinc smelters. Special considerations need to be provided for hot gases, presence of SO_x, high dew points, and sometimes for fire prevention Efficiencies are adequate for present emission standards, ranging 98-99 wt.%, and a clear stack is usually obtained. Restoration of permeability is usually by shaking alone. Air-to-cloth ratios are usually quite conservative because dust cake permeability is low. Dust is somewhat difficult to handle physically; problems of employee exposure to toxic materials are difficult to control. Possible means of improving efficiency and combating other problems are discussed. A tabular summary of some operating data is presented.

GENERAL INDUSTRY DESCRIPTION

The ores which are the raw material for primary lead and zinc production are usually found as combined lead-zinc sulfides. The major separation of lead from zinc takes place in a flotation process at the mill, commonly located at or near the mine site. Both lead and zinc ore concentrates are usually processed in the same plant, or at least by the same company in a complex of plants. There are some instances of buying and selling concentrates, intermediate products, or residues; these arrangements shift from time to time depending on metals markets and other factors. Lead is usually produced in 100 pound pigs or one ton blocks of metal, and zinc is usually produced in the form of zinc oxide or metallic zinc. Metallic lead is currently produced only by pyrometallurgical processes, whereas metallic zinc is produced by electrowinning followed by remelting and casting.

Lead and zinc smelters and refineries are large, capital intensive establishments, most of which rank in size only below integrated steel mills and copper smelters. The lead-zinc smelting industry in this country is old, much of the installed capacity having been built around the turn of the century and the most recent plants in the new Missouri lead belt having been built in the middle sixties.

Raw material sources may be categorized into two groups, the Missouri lead-belt ores and the Western ores. The Western ores are relatively high in silver, copper, cadmium, and antimony, and are highly variable according to specific mine source. The Missouri ores are much lower in these other metals, with copper being the major contaminant.

Because the raw material sources are non-uniform with respect both to time and location, and because different smelters may handle different sources from time to time, the industry is not highly attracted to a sophisticated level of automation and mechanization. Most process steps are either batch or continuous-batch so that time is available between process steps for various necessary changes in process parameters or reagent additions to permit the necessary metallurgical control. Only as the final steps in the process are approached does one find an opportunity for handling a reasonably standard, uniform product such as metallic lead, metallic zinc, or zinc oxide. Uniformity of process streams practically never occurs with the by-products, because these are the streams that provide the "balloon" for necessary changes in producing the primary products.

The nonferrous smelting industry is sometimes accused of being "backward". However, in view of the circumstances--high capital investment, large and uncontrollable variations in raw materials, and immediate and direct impact of world metal markets, a tendency to go slow is understandable. It is my impression that there is actually a high degree of technology involved in primary nonferrous smelting, but it is a different kind of technology than one finds in an automobile assembly plant or a plant producing synthetic chemicals.

There are only six primary lead smelter plants in the United States, and although I do not have an exact count, a correspondingly small number of primary zinc producing plants. Although the basic types of processing are usually quite similar, the number of plants is not sufficiently large to permit arriving at a "normal" or "standard" description of processes and related air pollution control facilities.

GENERAL FABRIC FILTER APPLICATION

In subsequent sections, tabular data is presented showing the typical range of application of fabric filters to smelter operations. However, some aspects of fabric filter application are rather common throughout and can be described in a single discussion. It will be obvious when the tables are studied that much data is missing; much of the information was collected incidental to other work, and much of the missing data probably has never been determined.

One of the most notable omissions is particle size information. Although this is regrettable from a scientific point of view, for practical applications it is not too important. Virtually all the processes generate emissions which are mostly fine metallurgical fume which would be expected to, and does, have a mass median aerodynamic diameter of one micron or less. If the fume is zinc oxide, it agglomerates readily so that if sufficient time is allowed, the end product is more like a mechanical dust than a fume. Of greater practical importance is the fact that the fume is "difficult"; that is, the permeability of the accumulated filter cake is low. The collected dust is, in general, somewhat difficult to handle, somewhat sticky, etc., as distinguished from a free-flowing granular material.

It will be noted that all the filter applications involve very conservative air to cloth ratios in order to achieve differential pressures across the fabric in the normally expected range, not exceeding 5 inches w.g. Differential pressures across the fabric are usually in that expected range, although accurate data are not available due to the large variations in operating conditions, which are commonly not measured at the same time that the pressure drop is monitored.

Efficiencies are usually in the range of 99 wt.% or better and seldom as low as 98%. The stack is usually visually clear and the outlet grain loading comfortably below present emission standards, whether official or unofficial, of about 0.02 grains per cubic foot. The efficiency is usually somewhat less than the 99.9+% that can usually be expected for a fabric filter under ideal conditions, for several reasons. One reason is that the baghouses are usually quite large, and in view of the high temperatures and noxious conditions, finding and fixing small mechanical leaks through the fabric is difficult and is not actively pursued as long as the emissions are satisfactory. Another factor is that, due to the very large square footage of fabric filter required, optimizing efforts in the past have usually been directed toward bag life, cost, and pressure drop with the fine points of increased efficiency taking a somewhat secondary role.

The hot gases from the furnace may be cooled to suitable baghouse temperatures by any of the classical methods. If acid gases such as SO_2 are present, the most favorable design would be to use radiant coolers to avoid the introduction of additional water vapor. When water sprays are used, they are usually used only for partial cooling, the remainder being by dilution in order to avoid too high a dew point.

If acid gases are present in significant quantity or if water spray cooling is used, the fabric filter must be kept above the dew point. Keeping the hopper above the dew point requires provisions over and above that required for the main housing. The collected bed of dust in the hopper provides an insulating effect so that the metal temperature may be far below the gas temperature. However, the vapors will still migrate through the bed of dust, condensing on the cold metal walls and forming a cake or mud.

Smelter baghouse testing has been conducted by EPA contractors. Such testing has focused almost entirely on quantifying the emissions from the baghouse and provides little or no design or operating data concerning the baghouses themselves or the inlet loading conditions. This limitation on such projects may or may not be appropriate, depending on one's point of view. However, one aspect of such testing is of prime importance.

In at least one such test with which we are familiar (1), the stack sampling was stopped during and for a short period following the shaking of a section in the baghouse, even though it was noted that a visible plume persisted for a significant fraction of a minute after the shake cycle. Obviously taking the data in this fashion resulted in a lower measured emission than would be obtained if the shake cycle were to be included. While this may, at first thought, be gratifying to the smelter manager, it in fact represents a strong potential for inequity in regulations. Regulations seeking "best available control technology" or its several variations would naturally be based on the data furnished to EPA. Thus the potential exists for the regulation to be based on a technology which is, in fact, somewhat less effective than the test data represent. One possible compensation for this bias would be to formulate emission standards based on the same testing methodology, but there are two objections to this. One would be the obvious, that such testing doesn't best represent the actual emissions; and the other would be that, in many installations, programming automated stack monitoring equipment in such a fashion would be difficult or impossible.

Fires in Metallurgical Baghouses

Fires are not uncommon in metallurgical baghouses. They usually but not always occur in the hopper of the baghouse. However, the circumstances under which they occur are highly variable, and immediate investigations following a fire frequently lack the application of scientific expertise. As far as we know, no concerted research has been done on the subject.

Fires occurring in the hopper are almost always caused by the presence of finely divided unoxidized metallics of a pyrophoric nature. Metallic zinc is probably the most common source of such fires, and metallic lead is probably the second most common source.

Prevention of fires in baghouse hoppers has been most successfully accomplished by the addition of an inexpensive diluent material. The most common diluent is agricultural lime. It obviously serves as a diluent to the combustible materials and also, if a fire does start, heat will be absorbed in dehydrating the lime thus furnishing an additional cooling effect. Some installations have used various carbonates such as soda ash or ground limestone, the theory being that the heat of the fire will release carbon dioxide which will tend to smother the fire. This latter action probably would not be effective for a fire in the bags since the filtering air flow would rather quickly flush the blanket of carbon dioxide away from the combustion area.

There is an empirical test called a "burning test" which is applied to controlling fires in metallurgical baghouses (2). In general the test consists of taking a shallow pan full of the collected dust and attempting to light it at one end by ordinary means such as a match or a burning paper towel. The time of burning from one end to the other of the pan is then measured, and if it is less than some specified value, more diluent is added to the duct entering the baghouse. The test is strictly empirical and the parameters need to be developed for each individual plant or baghouse situation.

One plant used a pan one-half inch deep, 2 inches wide, and 6 inches long. In this case, the burning was not actually timed; if the dust burned at all, more diluent was added. The diluent was hydrated lime (agricultural lime) and was added in amounts varying from 5% to 20% of the total dust load.

In another plant the test pan was 1 inch deep, 3 inches wide and 9 inches long. If the length of time required to burn from one end to the other was greater than 2 minutes, it was judged acceptable; if less than 2 minutes, agricultural lime was added to the flue.

Fires starting on the bags rather than in the hopper are usually blamed on the carryover of sparks although there are some indications that these fires also are sometimes started by metallics. In one interesting case (not a primary smelter), the bags of a reverse pulse filter with bottom inlet caught fire at the top of the bags and there was no fire in the hopper. Because at times sparks had been observed in the flue, the fire was blamed on sparks. It is difficult to understand how such sparks would preferentially migrate to the top of the bags. Another possible explanation would be that the reverse pulse air, having more energy and physically agitating the dust cake more at the top of the bags than further down, may have started the fire in the metallics present. In any case, the provision of a spark trap such as a cyclone or a spray chamber would prevent fires from sparks if they cannot otherwise be prevented. If both sparks and unoxidized zinc or lead are present, both types of precautions may be required.

Dust Handling

The handling of recovered dust in most secondary smelters is a difficult problem because such plants usually do not have a multiplicity of options for disposal of such material. The material may also have a significant monetary value. Its disposal by means other than by return to the process would be a serious problem in solid waste disposal.

The dust is usually somewhat difficult to handle. It does not flow readily from hoppers, etc., unless special provisions are made. Pneumatic conveying should be an applicable technique, but some operators have experienced difficulty with this operation due, in our opinion, to inadequately designed systems. Screw conveyors work, but they must be enclosed and dust control applied. Belt conveyors or other dusting type of material handling equipment is to be avoided.

The chemical composition of the dust is usually significantly different from the parent material and metallurgical considerations may govern the point in the process to which it is returned, or even if it is to be returned. Leaving aside the metallurgical considerations, dust returned in that form to a blast furnace, for example, would be mostly re-entrained and would immediately reappear in the baghouse, constituting a circulating dust load. The circulating dust load may be an operating advantage if such recirculation accumulates unwanted impurities so that a bleed-off or programmed disposal of the dust can be made. Otherwise, the dust will be recovered more efficiently by pugging, pelletizing, or briquetting of dust, or by incorporating it in sinter feed materials.

Baghouse Design

Considerable variation is found in baghouse design features, involving the basic type of structure, whether the baghouse is enclosed or insulated, whether there is a one-fan or a two-fan system, and whether section dampers are single or double. Almost all the baghouses applied to primary nonferrous smelting to date are of the shaking bag type with minimal application, to date, of pulse jet or reverse flow types.

The basic structure of the baghouse may be literally a building, constructed of brick or monolithic concrete, within which the cell plate, bags and shaking mechanism are located. The other extreme is the structural steel baghouse with sheet metal panels, insulated or uninsulated, and with or without auxiliary heating of hoppers. Many older baghouses are of the building type construction, but such construction is relatively expensive as compared to the other types. The relative merits of reliability, durability, problems of preventing condensation, damage by fires if such occur, are reasonably obvious.

Another design variable, usually associated with the era and type of basic construction, is whether or not the dust is handled in a cellar or by hoppers and conveying equipment. The cellar type of construction has the advantage that the collected bed of dust may be deliberately ignited so as to reduce any fire hazard in the following handling operations, and to densify the dust. As far as we know, such deliberate ignition has not been used for hopper and conveyor type systems. Problems with the cellar type of construction revolve mostly around the exposure of the front-end loader operators to toxic dust, and the creation of fugitives by the handling of the collected dust in this manner. These disadvantages could be overcome by providing a greater height of the cellar so that front-end loaders with pressurized clean air cabs could be used in them, and dust control can be visualized for subsequent handling of the material from the front-end loader into hoppers, trucks, or cars. All such provisions are fairly expensive, high labor cost solutions, so that all factors considered, it is doubtful that any more cellar-type baghouses will be constructed in the industry.

Without making a specific cost evaluation, it would appear that the scheme of constructing an ordinary baghouse inside a secondary structural steel and siding building offers the best overall solution in climates where condensation is a problem. Little or no supplemental heating is required for the surrounding building because most of the heat is furnished from the baghouse itself. Maintenance on conveying equipment, etc., would be conducted inside, minimizing fugitives and providing a better opportunity for dust control if necessary.

Some metallurgical baghouses have a two-fan system, one on the dirty side and one on the clean side of the baghouse. This permits the baghouse enclosure to operate at virtually zero pressure differential to atmosphere, which may be important in some cases. In general, the fan operating on the dirty side of the baghouse will suffer from increased erosion, corrosion, or caking and will be a high maintenance item. Further, in general, one blower can provide adequate static pressure for the system. Therefore, unless an unusually high static suction is required for the system or unless the structural problems are unusual, use of one blower would seem to be more cost-effective. If conditions are not unduly onerous for blower operation, it may be on the dirty side of the baghouse, otherwise it should be on the clean side. Locating the blower on the clean side of the baghouses means that it is under suction, resulting in fewer problems when it is necessary to enter a section for maintenance, but higher structural cost.

Some smelter baghouses are provided with section dampers on both the inlet and outlet of each section. These are usually found where the outlet gases from the baghouse go to a tall stack and the system fan is on the dirty side of the baghouse. The outlet damper is then necessary to stop the significant draft that would occur due to the stack height when the section is isolated for maintenance.

APPLICATION TO SPECIFIC SOURCES

Lead Smelters

The major sources controlled by baghouses in lead smelters are the sintering machine and the blast furnace. In many cases a dross reverberatory furnace is associated with the blast furnace and served by the same baghouse.

The sintering machine is similar in principle to the well-known sinter machines in the steel industry. Most lead smelter sinter machines are updraft units. First an ignition layer is placed on the grates, ignited, and then covered by a much thicker main layer. The sinter feed is basically ore concentrate, return sinter, recycled lead-bearing materials and perhaps additives to promote slagging later in the blast furnace. The sinter machine off-gas typically is used as a feed gas to an acid recovery plant, and is desired to contain about 5% or more SO_2. The circuit of gases through the sinter machine may be somewhat complex, with weak gases produced toward the end of the machine, where SO_2 evolution is low, being recirculated through an upstream section of the machine to enrich the SO_2 content before going to the acid plant.

The sinter machine off-gases contain dust and fume and must be cleaned before being processed in the acid plant. This is typically accomplished either by fabric filter or by electrostatic precipitator. For fabric filters, the available data is presented in Table 1.

Blast Furnace

The product of the sinter plant is lump sinter, fused chunks of lead oxide, lead, slag and other impurities. The sinter is charged into the blast furnace with coke as a reductant and with more fluxes. The blast furnace off-gas contains a lead oxide fume, dust, trace amounts of SO_2 and significant amounts of carbon monoxide. Most smelters control the emissions from the blast furnace by use of a baghouse. The same baghouse frequently serves the dross reverberatory furnace, where the dross from lead kettles is remelted to recover the contained metallic lead and to produce matte and/or speiss in a handleable form. The blast furnace gases typically go through a settling chamber or a long low-velocity flue to provide cooling and to settle out any sparks. Sometimes water cooling sprays are installed in the flue.

Table 2 shows the parameters involving the baghouses and their operations on blast furnace/reverb furnace duty.

The zinc fuming furnace is, in some instances, physically located in the lead smelter because it can be charged with molten slag from the blast furnace to recover the contained zinc as zinc oxide. However, since the product is zinc, and since the zinc fuming operation is not necessarily conducted in the lead plant, it is included under zinc plants.

ZINC PLANT

Zinc sulfide ore concentrates are oxidized in one of several possible styles of roaster. Since the desulfurized product is to be dissolved in acid and used as feed for electrowinning, there is no need to prepare a sinter product. The roasters may be a hearth type roaster or a combination flash and hearth type roaster. Off-gases from the roaster are rich in SO_2 and are usually the feed for an acid recovery plant. Either electrostatic precipitators or fabric filters are used to clean the off-gas for acid plant processing. Characteristics of fabric filters used for this purpose are shown in Table 3.

Zinc Fuming Furnace

In the zinc fuming furnace, metallic zinc vapor is evolved from the blast furnace slag and oxidized to zinc oxide fume. Radiant hairpin coolers or long low-velocity flues, or a combination of both, are used to cool the gas. The long transit time allows agglomeration so that the product, captured in a fabric filter, is more like a dust than a fume in its appearance and handling characteristics. Fabric filter applications in this duty are shown in Table 3.

TABLE 1

LEAD SMELTERS

SINTER MACHINE

	FABRIC FILTER ITEM NO.			
CHARACTERISTIC	7	8	9	10
F.F. Type	Shaker	Shaker	Shaker	Shaker
Fabric	-	Acrylic	-	Polyester
Temp °F	200	215	260	240
A/C Ratio acfm/ft^2	1.1	1.1	1.8	1.2
ΔP in.w.g.	-	3.4	4.0	-
Inlet gr/scf	2.0	1.4	2.1	-
Outlet gr/scf	-	0.001	0.003	-
Cooling	-	Water Spray	Water Spray	-
SO_2 %	5.5	-	-	5.0
Bag Life Mos.	-	36+	23+	-
Dust Removal	Screw Conv.	Cellars	Cellars	Screw Conv.

TABLE 2

LEAD SMELTERS

BLAST FURNACE

	FABRIC FILTER ITEM NO.					
CHARACTERISTIC	1	2	3	4	5	6
F.F. Type	Shaker	Shaker	Shaker	Shaker	Shaker	Shaker
Fabric	-	-	Wool	Acrylic	Acrylic	Acrylic
Temp °F	200	200	160	160	150	-
A/C Ratio acfm/ft^2	1.8	0.5	1.6	1.6	0.55	1.3
ΔP in.w.g.	2.0	2.5	3.5	3.5	2.0	3.5
Inlet gr/scf	2.0	-	1.1	2.6	0.9	1.24
Outlet gr/scf	0.05	-	0.008	0.006	-	0.01
Cooling	-	Infilt.	-	Water sprays	Infilt., water sprays	-
SO_2 %	0.1	-	-	-	-	-
Bag Life mos.	30	-	53	40	56	-
Dust Removal	Screw Conv.	Cellars	Cellars	Cellars	Cellars	Screw Conv.

TABLE 3

ZINC PLANT

CHARACTERISTIC	FABRIC FILTER ITEM NO. 11	12	13	14
F.F. Type	Shaker	Shaker	Shaker	Reverse
Fabric	Polyester	Nomex	Acrylic	Glass
Temp °F	300	300	240	500
A/C Ratio acfm/ft^2	2.1	1.8	1.06	1.96
ΔP in. w.g.	5.0	3.2	5.0	8.0
Inlet gr/scf	-	6.7	15.0	5-8
Outlet gr/scf	0.028	0.003	0.0055	0.05
Cooling	Radiant	Radiant	Radiant	WHB
SO_2 %	-	-	-	6.5
Bag Life mos.	-	25	30+	24
Dust Removal	Screw Conv.	Screw Conv.	Screw Conv.	Screw Conv.
Application	Zinc Fuming Furnace	Zinc Fuming Furnace	Zinc Fuming Furnace	Roaster

MISCELLANEOUS OPERATIONS

A few miscellaneous operations are shown in Table 4.

Fabric filters are not widely used in the primary smelting industry on miscellaneous dust sources but it is predictable that their use will become more widespread. In the industry, dust from material handling operations is distinguished from furnace ventilation by calling the former "mechanical dust" or "hygiene ventilation". For such applications, low to medium pressure drop wet scrubbers have been applied more frequently than have fabric filters. One reason for using wet scrubbers is that, in the sinter preparation and sinter return circuits, the materials being handled are often hot and moist, ruling against the use of fabric filters without extensive gas pretreatment. However, from both operational and hygiene points of view, it is coming to be regarded as a net advantage to handle such materials dry. Even though much more dust is generated when handling dry materials, the associated problems with duct work, housekeeping, etc., are much greater than if dry materials were handled. Well-designed dust control hooding can avoid excessive dust pickup while still preventing dispersion of the dust into the workplace air, and all other problems are simplified. For such applications, the use of fabric filters of course would give a higher efficiency or a lower emission than would the use of low or medium pressure drop wet scrubbers, and fabric filters are usually more cost-effective than high pressure drop scrubbers. A further consideration is the fact that many nonferrous smelters are located in areas where water supply is restricted.

MODIFICATION FOR IMPROVEMENT

As previously described, the rather difficult duty for baghouses on smelter furnace emissions, combined with the onerous working conditions inside, results in a somewhat lower degree of maintenance and correction of minor fabric leakages than is possible. There are several design changes that could be made to result in better efficiency performance.

It is not uncommon to see even the best of metallurgical baghouses with an inch or more of powdery dust on the cell plate on the clean side of the fabric. An improvement in collection efficiency could be achieved by use of more efficient media, and a philosophy of cleaning up the clean side of the baghouse and keeping it clean.

Reported experience with development projects for fabric filters in power plants offers a clue. Such installations have stack monitors for total particulate. It has been observed that, by correlating stack monitor readings with the cleaning cycle for different filter compartments, an early detection can be obtained of minor leakages. If the offending compartment is then shut down and inspected, it is easier to find the leakage since the "clean side" of the bags really is clean, and leakages can be found and fixed before they become serious.

TABLE 4

MISCELLANEOUS

CHARACTERISTIC	FABRIC FILTER ITEM NO. 15	16	17
F.F. Type	Reverse & Shake	Shaker	Shaker
Fabric		Acrylic	Acrylic
Temp °F	250	200	200
A/C Ratio acfm/ft^2	1.92	2.1	1.67
ΔP in. w.g.	-	4.0	4.0
Inlet gr/scf	-	2.6	2.6
Outlet gr/scf	-	-	-
Cooling	-	-	-
SO_2 %	-	1.56	1.56
Bag Life mos.	-	30	30
Dust Removal	Screw Conv.	Screw Conv.	Screw Conv.
Application	Dross Arc Furnace	*	*

* Weak end of sinter machine plus misc. sources

Along with the early warning system described above, it would be necessary to have a well-designed vacuum cleaning system so that the cell plate and housing can be cleaned following every maintenance operation.

Bag testing experiments could be made in which the emphasis is shifted more strongly to considerations of efficiency. Unfortunately, efficiency experiments cannot be done with one or a few bags in each compartment like experiments on bag life. Following preliminary small-scale experiments, a real operating experiment would require resuiting an entire compartment.

The design of fabric filters commonly provides enough sections so that the net filtering area is adequate when one section is off-line for cleaning. Some baghouses provide a second extra section so that the net capacity is adequate when an additional section is off-line for maintenance. This practice should be expanded, in order to properly compensate for the operational changes suggested above.

OSHA VS. EPA

For various reasons, government regulation of emissions and ambient air quality is compartmentalized in the Environmental Protection Agency, and government regulations concerning contaminant concentrations in the workplace are lodged with the Occupational Safety and Health Administration of the Department of Labor. This is unfortunate in several respects. Of major importance is the fact that contaminant concentrations inside the plant at the workplace, and contaminant concentrations in the stack or ambient air, are the two opposite ends of the same engineering problem. A second historical fact is that EPA got started first. Industry frequently compartmentalized itself to match the government agency structure. The two functional problems, similar as they are in technical content, thus meet only at the engineering function where equipment design and installation is involved. Since compliance with most EPA requirements is regarded as a constraint to be met with minimum cost, and since EPA requirements preceded OSHA requirements, there has been a trend in recent years to place engineering emphasis on EPA requirements; attention to the OSHA-type requirements was minimized or forgotten.

Now we have both requirements with us. There is a degree of tunnel vision on the part of both agencies regarding single-minded pursuit of their objectives without consideration of other government requirements, due largely if not entirely to the legalities and bureaucratic structure involved. As a result, there are many severe problems looming on the horizon. Some of the requirements are complementary to each other, as in the example given above for keeping the clean side of the baghouse clean. The steps described lead to both improved efficiency performance of baghouses and to reduction of exposure to the workers doing the baghouse maintenance. That fortunate compatibility does not exist in another and very important area.

EPA, as you all know, is beginning to emphasize fugitive emissions from the smelter, as well as stack emissions. With a stringent fenceline lead-in-air standard looming, it is predictable that there will be major emphasis on controlling fugitive emissions. All of you who have had an opportunity to contemplate this problem realize its magnitude. Fugitive sources include the discharge of ventilation air from the smelter, the emission of contaminated air through roof monitors, etc., storage piles, roadways, etc.

One technique that has been attempted, in focusing on EPA requirements, is what I call building containment. The entire building is enclosed and the roof monitor exhausted through a fabric filter and up a tall stack if necessary. If one can live with the results of such an installation, it does solve the problem of fugitive emissions from the smelter building. Such installations are very expensive, requiring perhaps as much as one million cfm capacity, in an attempt to emulate mother nature's ventilation. The baghouse is so big that it usually must be located remotely, so that long, large and expensive flues are required, the power requirement is significant, etc., etc. For these reasons, if EPA compliance is all that's in the designer's mind, the capacity of such an installation is cut to the bone and working conditions inside become significantly worse.

Two such building containment installations have been made in the copper smelting industry, both of which are serious failures in terms of internal working conditions. Both installations were designed on an "air changes per hour" basis which, aside from the fundamental question of whether such an installation is appropriate in the first place, is the poorest design basis that could be selected. Designing general ventilation on the basis of air change rates results in a capacity related to the size of the building rather than to the size of the problem. While that approach may be appropriate for some relatively standard situations like school classrooms, where the students can be packed only so close and the ceiling height ranges only between 8 and 12 feet, it does not apply to most industrial operations since there is such a wide variation from one plant to another. This variation is even more extreme in the nonferrous smelting industry than in most others.

Proper design of a building containment system must be based, as a minimum, on an attempt to remove hot contaminated air at least as effectively as occurred with natural ventilation before the containment. This means estimating the velocity and volume of thermal plumes, which is exceedingly difficult in view of the size of the operations and their irregular nature. Because workplace conditions seldom meet OSHA standards now, general building containment ventilation based on this concept would still be inadequate, but at least the conditions inside would not be made worse than they are now.

Many of the sources in the smelter can be controlled locally, although in view of the nature of the operations, extraordinary effort may be required, and more than one try may be required, to achieve success. Assuming that success is eventually achieved by local control, the payout is significant.

The potential payout assumes one of two forms. It is possible, but not highly probable, that major advances in local control of fume and dust would make the roof monitor emissions of the smelter building so low as to escape the need for further control. That would indeed be a happy state of affairs. A much more likely result is that the local control would reduce the contaminant burden of the containment air sufficiently so that air cleaning equipment much less expensive than heavy duty fabric filters would be adequate. The particulate collected in the containment systems mentioned amounts to about 100 lbs/hr. With better local control of sources, this rate could be reduced to the point where much less expensive filtering equipment, or perhaps none at all, would suffice for the containment system. Much higher general ventilation rates would then be affordable. The combination of local control and copious general ventilation would improve, rather than degrade, working conditions inside the building.

Assuming that good faith efforts on the part of the industry can be established and recognized, it seems only reasonable to ask both EPA and OSHA to be cognizant of each other's impact on the industry. The agencies should refrain from rule-making incompatible with the other agency's objectives and rules. The agencies should recognize the increased cost and time required for the industry to cope with both sets of requirements, in what is essentially two artifically defined parts of the same engineering problem.

ACKNOWLEDGEMENT

The kind assistance of the many individuals in the industry, who furnished much of the tabulated data, is sincerely appreciated.

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REVIEW OF DESIGN AND OPERATION OF ELECTROSTATIC PRECIPITATORS IN NONFERROUS METALS APPLICATIONS

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ABSTRACT

Electrostatic precipitators for applications in the nonferrous metals industry belong in a group representing one of the earliest applications for electrostatic precipitators. For example, White[1] lists several "firsts"; Lurgi, one of the major suppliers of electrostatic precipitators worldwide, started with an electrostatic precipitator for a Herreshoff furnace in 1913. Despite this long history, the application is not considered easy and free of risks. The diversity of processes and equipment used in this industry has made every installation a unique system and a challenge for its designers and operators.

This paper is written with the intent to provide a review of some practical design considerations, applications, and operating experiences with electrostatic precipitators in the nonferrous metals industry, specifically for applications in primary smelters for copper, zinc, and lead.

This paper is based on information provided by Lurgi based on 65 years of service to this industry.

Review of Processes and Electrostatic Precipitator Applications

The selection of methods to separate dusts from waste gas streams and to recover entrained metals is largely dependent on the process, i.e., type of smelter and the particular equipment, such as roaster, converter, dryer, etc., and on the raw materials used in the process. A further complication is caused by the sulfur oxides generated at various stages, which not only require a high degree of dust collection prior to their own removal from the waste gas stream, but which also to a large extent determine and complicate the dust collection system. Costs of gas cleaning systems depend basically on the gas volume to be treated and increase progressively with the required collecting efficiency.

A review of various gas producers in nonferrous metal smelters was made by Wanner[2]. Table 1 shows typical gas producers and recommended dust collectors in copper, zinc, and lead smelters. Dust collectors for waste gases in nonferrous metal smelters are:

- electrostatic precipitators
- fabric filters
- scrubbers
- cyclones

Operating conditions may limit collectors to specific applications. In general, dust collectors have to be designed for continuous heavy-duty operation, a condition which again reflects itself in the cost of such equipment.

So-called hot-gas electrostatic precipitators are used to collect dusts and to a certain degree fumes from waste gas having an appreciable SO_x content. Because of the sulfuric acid dew point around 150 to 220°C., operating temperatures of 300 to 400°C. are normally selected. This applies to precipitators for reverberatory furnaces, electro-furnaces, flash furnaces, dryers, converters, and roasters.

Fabric filters with glass fabrics with silicone finish can be used for gas temperatures up to 250°C. Their application is limited because of the high sulfuric acid dew points and H_2F_2 compounds in some waste gases. With sticky dust, cleaning of the fabric filters proves to be difficult, but fabric filters offer a definite advantage where extreme low residual dust contents, for example, less than 10 mg/m^3 NTP are required.

Scrubbers are often used for the cleaning of waste gases in this industry; for example, venturi scrubbers in various designs. Energy requirements increase with the pressure drop through the scrubber, but for fine particles, high pressure drops are required to achieve an acceptable collecting efficiency. For example, the pressure drop of a Venturi scrubber for the separation of zinc oxide particles of 0.5 micrometer diameter is approximately 2,000 mm W.G. for a collecting efficiency of 96 percent. The main advantage of scrubbers are their low investment cost; the main disadvantage are the need for sludge processing equipment and corrosion-resistant material.

Cyclones are used in cases where particles are larger than 5 micrometers and collecting efficiencies are low. Particles larger than 30 micrometer can be collected at almost a 100 percent level. In cases of extremely high dust loads, cyclones are often used as pre-collectors. Advantages of cyclones are simplicity of design and capability of handling gases up to 1000°C., besides low investment costs and small space requirements. Their main disadvantage is the rather limited collecting efficiency for fine particles.

The traditional design of wet-process precipitators is the tube or plate-type lead precipitator, but lead as a material of construction for the precipitator; i.e., shell and internal parts, has disadvantages, and wet-process precipitators made out of plastics have been developed. Lurgi has

Table 1. GAS CLEANING IN NONFERROUS SMELTERS

SOURCE OF GAS	GAS TEMPERATURE °C.	SO_2 CONTENT %	DUST CONTENT g/Nm^3	RECOMMENDED DUST COLLECTOR
COPPER SMELTER				
Converter	300-400	5-9	2-9	Electrostatic Precipitator (ESP)
Reverberatory Furnace	250-350	1-1.5	5-10	ESP
Flash Smelting Furnace	350	8-11	30-80	ESP
Concentrate Dryer	100-120	.2	100-200	ESP
Fluid Bed Roaster (Partial)	400	13-14	1,000	Cyclone, ESP
Fluid Bed Roaster (Dead)	300	8	300	Cyclone, ESP
Electro Furnace	350	1-6	30-50	ESP
Slag Furnace	500-1,000	.5	5-10	Scrubber
ZINC SMELTER				
Fluid Bed Roaster	350	9-11	100	Cyclone, ESP
Sintering Machine	300-400	5-7	20-30	ESP
ISP Furnace	350	-	-	Scrubber
Waelz Kiln	350	-	50-100	ESP, Fabric Filter
LEAD SMELTER				
Sintering Machine	300-400	5-7	20-30	ESP
Shaft Furnace	150-250	-	10	ESP, Fabric Filter

supplied more than 120 electrostatic precipitators made mainly out of plastics or with plastics used for major components. Compared with lead precipitators, plastic precipitators offer considerable advantages such as:

- design suitable for high negative pressures
- low maintenance and repair costs
- modular design
- low erection costs
- maximum continuous operating temperature of 80°C.

Design of Electrostatic Precipitators

The basic design of hot-gas precipitators is similar to the standard design of dry-process precipitators used in other industries, except that the design used in nonferrous metal applications is suitable for gas temperatures up to 450°C. If the gas does not contain any components causing corrosion, normal carbon steel can be used in this temperature range since its scaling rate is minimal as long as the gas contains less than 5 percent O_2.

Electrode systems consisting of discharge wires and collecting surface plates are grouped in single or multiple electrical fields in series or parallel, installed in single or double-chamber casings. Precipitators are built with horizontal or vertical gas flow, the former restricted to plates as collecting surfaces and the latter normally restricted to a single field design, but available as plate-or tube-type precipitator.

Collecting surface plates oppose the discharge wires in the electrical field; electrically they are connected to the positive outlet of the power supply and grounded with the precipitator casing. Their support structure can be part of the casing. Collecting surface plates form individual gas passages (Figure 1) with discharge wires centered between rows of plates, resulting in a generally non-uniform electrical field.

Collecting surface plates are made out of cold-rolled steel sheets normally 18 to 24 in. wide and up to 45 ft. high. Assembled they form curtains up to 18 ft. long. Roll-formed profiles offer rigidity, pockets with little or no gas flow, and good vibration characteristics. Wet-process precipitators use a simplified collecting plate design in heavier gauge material.

Tube-type collecting surfaces offer a uniform electrical field but only the inside can be used as collecting surface; hexagonal, square, or segmented collecting surfaces allow the use of both sides, but lose some of the uniformity of the electrical field of round tubes. Wet-process precipitators use collecting surfaces made out of lead or plastic, in tube-or plate-type design.

The discharge system, including its support structure, is connected to the negative outlet of the high-voltage power supply. Discharge wires are either suspended from a horizontal support system or individually mounted in vertical frames. Dry-process precipitators use rapping systems to keep the discharge wires clean; spray systems in wet process precipitators clean both collecting surfaces and discharge wires. Discharge wires with round, square, or star-shaped cross-sections (Figure 2) are used. Spiked discharge wires

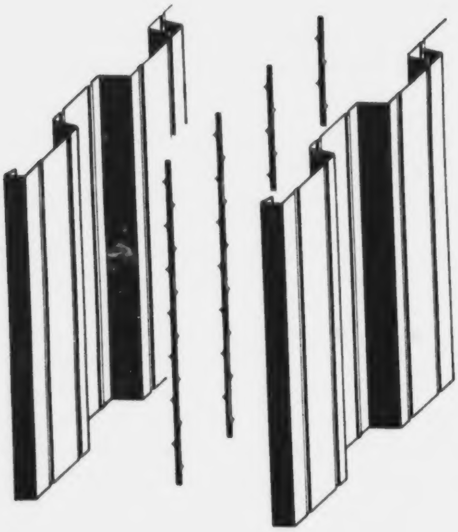

Figure 1. Collecting Surfaces

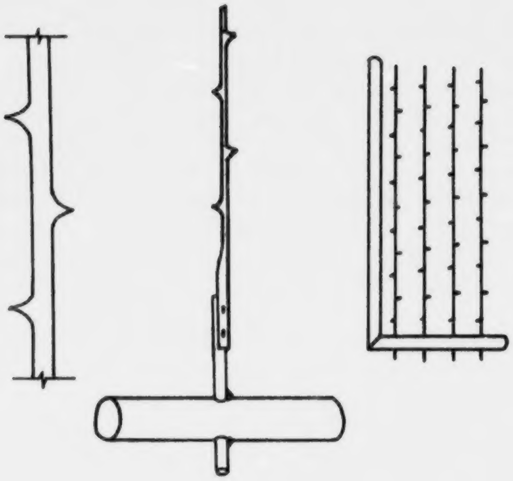

Figure 2. Discharge Wires

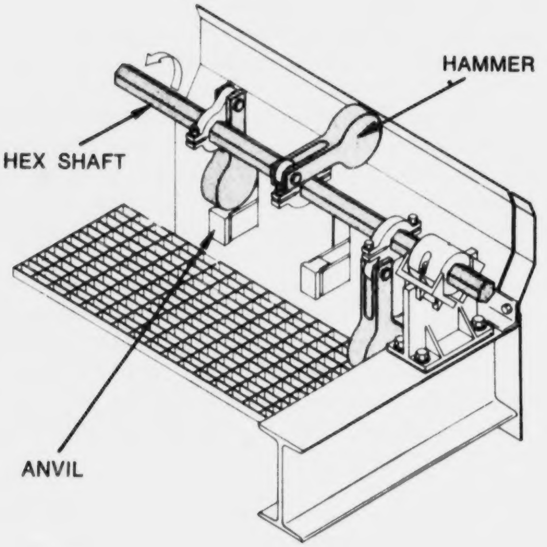
HAMMER
HEX SHAFT
ANVIL

Figure 3. Collecting Surface Rapper

offer higher currents at somewhat lower voltages. Wet-process precipitators for sulfuric acid mists use star-shaped lead discharge wires with a steel core for strength. Special wires with spikes have been developed for precipitators with square tubes as collecting surfaces. Removal of precipitator dust is done with rapping systems in dry-process precipitators, and with continuous or intermittent spray systems in wet-process precipitators. Various mechanisms enter into the removal of the precipitated dust from collecting surface plates by rapping, the most important being the acceleration of the plate normal to its surface, commonly measured in g ($1g = 9.81\ m/sec^2$). Experience dictates a minimum of 200g at any given point of the collecting surface plate. Periodical rapping of the collecting surface plates is achieved by rotating hammers impacting on the anvils of the guide bars at the bottom of the collecting surface plates (Figure 3).

Wet-process removal of the accumulated dust by flushing requires a good distribution of the water jets over all of the collecting surfaces, including the discharge system. Continuously operating spray systems may cause excessive arcing on top and bottom of the electrical field. Discharge wires mounted in vertical frames are kept clean by an effective rapping system using rotating hammers (Figure 4); rapping accelerations have been reported by Engelbrecht[3]. Frames are made out of square or tubular pipes with suitable cross-sections for structural strength.

The discharge system is supported and electrically isolated from the precipitator casing by four support insulators for each electrical field. Insulator materials are porcelain, alumina, or quartz. The insulators also serve as a seal between the inside of the precipitator and the ambient air. Heaters are used to prevent condensation on the surfaces of the insulator during start-up and shut-down periods. Purging systems with heated air or clean gas are used in special cases to provide additional protection against condensation.

Electrode systems and their supports are reinforced to account for the possibility of dust build-ups; also reinforced are the rapping mechanisms and the dust handling systems due to the possible stickiness and abrasive characteristics of the dust and clearances outside of the electrical field are increased to allow for dust build-ups without causing short circuits.

Normally, electrode systems, i.e., discharge wires and collecting surface plates are enclosed in the "smallest possible" housing; in some cases, additional access and space for manual cleaning has to be provided. Additional rapping systems can be used for the hopper walls. Monitors for the continuous operation of the dust discharge system and the dust temperature are recommended. Dusts can be pyrophoric; continuous operation of the electrode rapping systems should be monitored.

The precipitator casing encloses and supports the electrical field(s). Casings are designed as self-supporting structures using steel plates with outside stiffeners as supporting elements, they are welded gas-tight and thoroughly insulated against heat losses. Inlet and outlet gas ducts are connected to the precipitator casing with transition pieces, incorporating gas distribution devices, such as vanes, baffles, or perforated plates. Various designs are used with good success (Figure 5). Minimum requirements

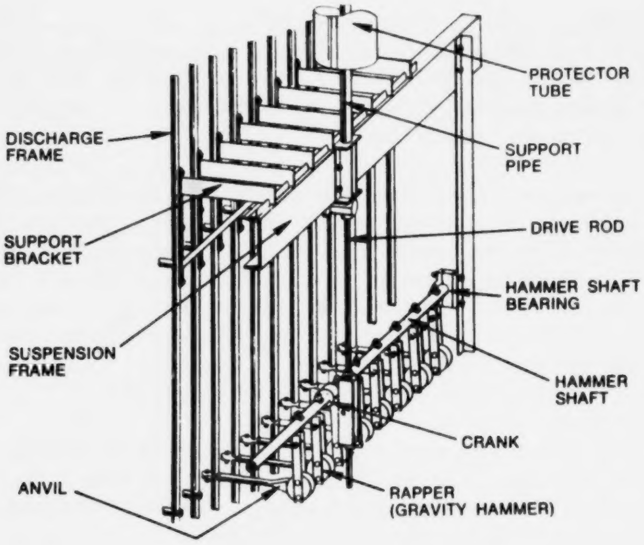
PROTECTOR TUBE
DISCHARGE FRAME
SUPPORT PIPE
DRIVE ROD
SUPPORT BRACKET
HAMMER SHAFT BEARING
SUSPENSION FRAME
HAMMER SHAFT
CRANK
ANVIL
RAPPER (GRAVITY HAMMER)

Figure 4. Discharge Wire Rapper

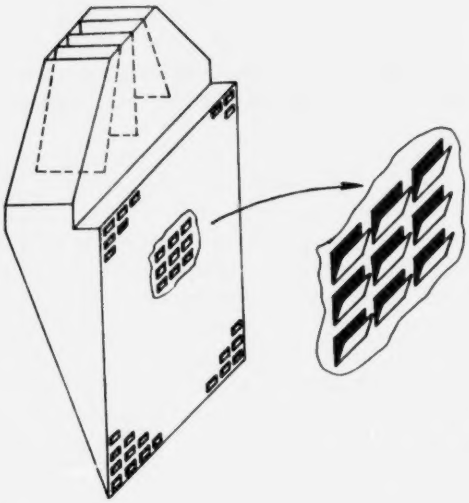

Figure 5. Gas Distribution Plate

for gas velocity distribution have been set by the Industrial Gas Cleaning Institute[4]; flow model test results have been reported by Engelbrecht[5].

Rapping systems for gas distribution plates are required for all dry-process precipitators in this industry. Drag conveyors to evacuate the dust through hoppers are used; air inleakages through hopper air locks, inspection doors, drive shaft seals, etc., have to be kept to an absolute minimum to prevent severe corrosion problems.

High-voltage power supplies are not different from those used for other precipitator applications. Automatic voltage control systems are designed to keep the precipitator voltage below the fluctuating arc-over voltage, to reduce the voltage immediately after an arc-over inside of the precipitator and restore the operating voltage immediately after arcing. The last two requirements minimize the periods of reduced collecting efficiency and prevent damage to the precipitator by electro-erosion. Automatic voltage controls have been developed to satisfy these requirements (Crynack[6]).

Wet-process precipitators require that the gases entering the precipitator be saturated; a wet-scrubber can be used for this purpose. Dust particles and mist droplets are precipitated together on the immersed collecting surfaces forming a thin slurry. To maintain a free flow of this slurry, it is recommended to keep the dust burden at the inlet of the precipitator below $1g/m^3$. The collecting surfaces and the discharge system are flushed periodically with the precipitator de-energized in this section. Due to the lack of particle re-entrainment, wet-process precipitators can achieve high collecting efficiencies. Lead has been a time-proven material of construction for sulfuric solutions in an atmosphere containing oxygen. The shell of the precipitator is made of lead sheets supported by a steel structure. The collecting surfaces (plates or tubes) are made out of lead; all other components of the electrode system are made out of lead-clad steel parts. Lead is not always sufficiently corrosion proof, especially if substantial amounts of halogenides are present. Neither are the mechanical characteristics of lead satisfactory for long time use.

Therefore, it became necessary to develop an alternative design using fiberglass reinforced plastic as material of construction (Figure 6). Approximately 120 plastic precipitators have been built by Lurgi during the last 20 years. The collecting surfaces, located between the upper and lower housing, are made out of non-conducting plastic. The electrical conductivity of these collecting surfaces is provided by a uniform liquid film (Figure 7). Special consideration has to be given to the grounding of the collecting surfaces. The discharge electrodes are either lead-clad steel rods or made entirely out of high-alloy steels. Spiked electrodes are used extensively.

Precipitator casings are also made out of plastics using, for example, a glass-fiber reinforced polyester resin with a PVC liner.

Since plastic is rather expensive as a material of construction, it is important to use all advantages this material offers to the fullest extent.

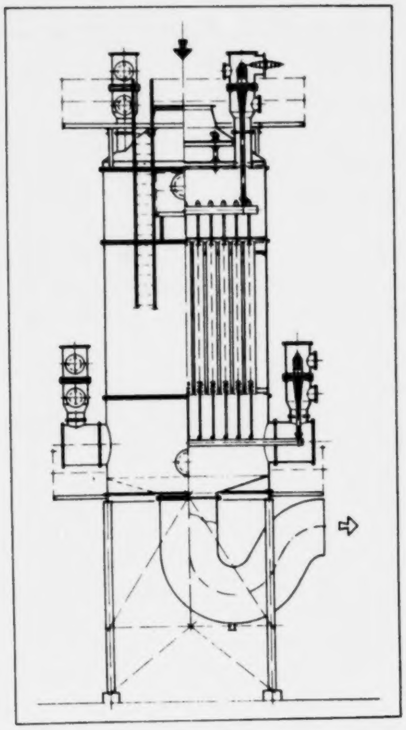

Figure 6
Plastic Precipitator
(Schematic)

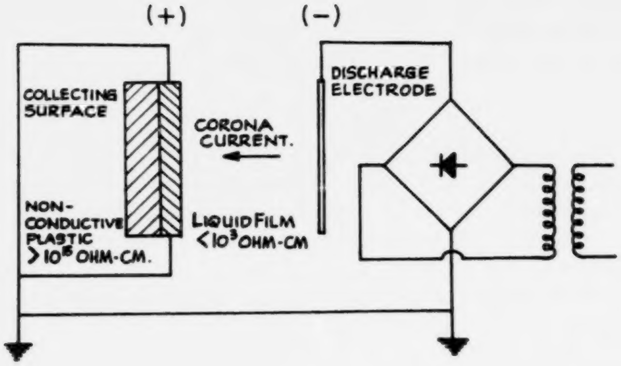
(+)
(−)
COLLECTING SURFACE
NON-CONDUCTIVE PLASTIC >10^15 OHM-CM.
CORONA CURRENT.
LIQUID FILM <10^3 OHM-CM
DISCHARGE ELECTRODE

Figure 7. Collecting Surface (Schematic)

Operation of Electrostatic Precipitators

The continuous operation of an electrostatic precipitator at a high collecting efficiency level depends normally on:

- adequate size
- suitable design
- proper operation
- good maintenance

The first two conditions have to be met by the supplier. Proper operation and maintenance procedure can be established by the plant operator using information and manuals provided by the supplier, past experience or information provided in the literature, (Englebrecht[7]).

The dust collection process in an electrostatic precipitator starts with the charging of particles, their transportation to, and precipitation on the collecting surfaces. In this process, particles will be deposited on top of other particles causing the formation of a substantial dust layer on the collecting surfaces. This causes no serious problems as long as the dust particles are of a relatively low specific electrical dust resistivity which allows for the electrical charge to bleed off to the grounded collecting surface. However, in the case of a high resistivity, for example, in excess of $2x10^{10}$ ohm-cm, these charges cannot be reduced quickly enough and a layer of charged dust particles forms on the collecting surfaces (Figure 8). This will eventually lead to a reduction of the voltage gradient between the discharge electrode and the outmost dust layer; and consequently, to a reduction of the precipitator collecting efficiency (Wanner[2], Figure 9).

It is important for the supplier of the electrostatic precipitator to be aware of the dust resistivity level, which typically depends on the gas temperature (Figure 10). The upper curve exceeds the critical level of $2x10^{10}$ ohm-cm within a certain temperature range whereas the lower curve representing a carrier gas with an SO_3 content of 3 mg/m^3 is completely below the critical level. The dust resistivity can be reduced by conditioning, i.e., by adding certain condensable substances such as SO_3, NH_3, HCL, Na_2CO_3. Conditioning by water injection into the process is also feasible (Eisert[8], Figure 11).

An SO_3 level of approximately 20 ppm is normally sufficient for conditioning. For dusts with a high basicity, for example Pb-Zn fumes, SO_3 levels up to 200 ppm may be required. Conditioning with SO_3 is often achieved by generating SO_3 in the system itself, provided the gas has sufficiently high SO_2 and O_2 levels and dust oxides are available as catalysts. The process requires that a conversion temperature of 420 to 800°C. is maintained for a sufficient period of time.

Problems with Electrostatic Precipitators

As much as SO_3 conditioning seems to be beneficial for the operation of the electrostatic precipitator, a high SO_3 level can cause a series of problems, such as:

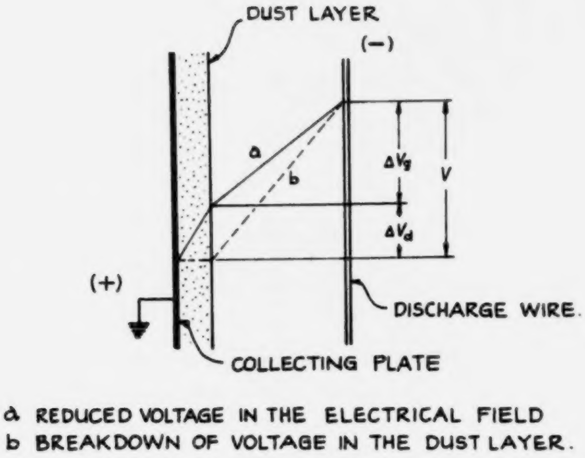
DUST LAYER
(−)
a
b
ΔV_g
V
ΔV_d
(+)
DISCHARGE WIRE.
COLLECTING PLATE
a REDUCED VOLTAGE IN THE ELECTRICAL FIELD
b BREAKDOWN OF VOLTAGE IN THE DUST LAYER.

Figure 8. Effect of dust resistivity

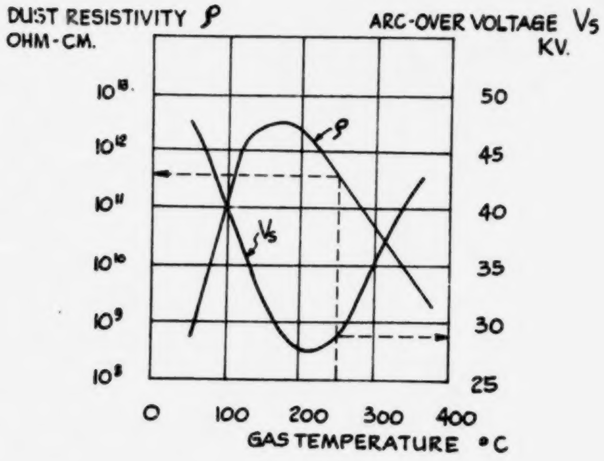

Figure 9. Dust Resistivity and Arc-Over Voltage

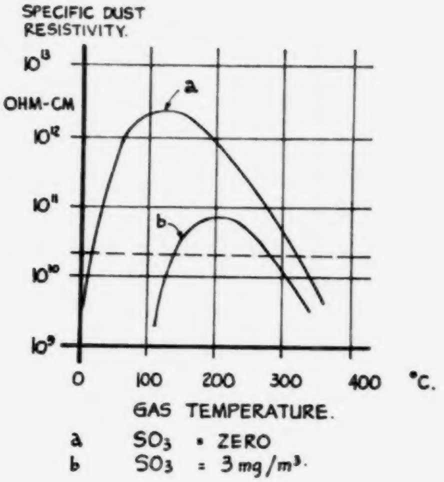

Figure 10. Dust Resistivity as a function of gas temperature and SO_3 content

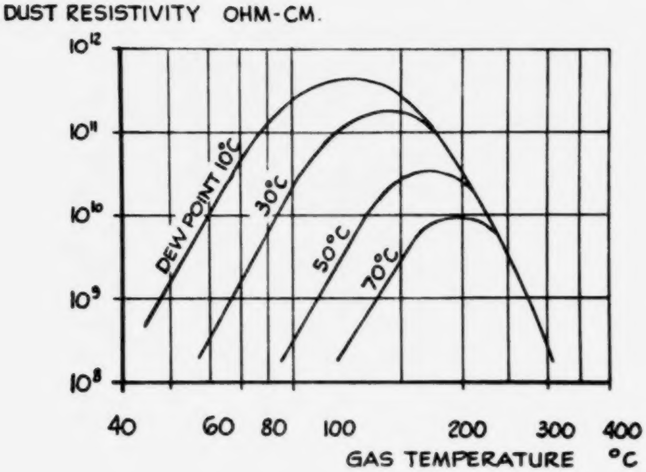

Figure 11. Dust Resistivity as a function of gas temperature and dewpoint

- increase in corrosion rate by increase of the sulfuric acid dew point.
- SO_3 can sulphate the dust, making it sticky and difficult to handle.
- the size of the mist precipitator in the acid plant has to be increased.

Thus, the rate of cooling, i.e., the residence time in the temperature range between 420 and 800°C. has to be carefully considered.

An example of an electrostatic precipitator for waste gases from a zinc roaster is used to explain a major problem of precipitator applications in this industry, namely, the dust build-up on collecting surface plates and discharge wires.

The reaction in a zinc roaster generates SO_2 in the roaster, leading to SO_3 in the waste-heat boiler at temperatures below 820°C. The reaction depends on the presence of catalysts and drops to zero at gas temperatures below 420°C., resulting in SO_3 and zinc in the form of zinc sulphate. Normally, this does not necessarily lead to build-ups in the precipitator. But in a number of cases, a heavy build-up is occurring on both the discharge wires and the collecting surfaces; the mechanisms leading to these build-ups can be analyzed.

Dust build-ups consist of dust particles sticking on other dust particles sticking on other dust particles; thus, the conditions on the surface of the dust particles have to be investigated. Two different kinds of build-ups can be observed:

- in vicinity of the waste heat boiler, the cyclones, the roaster precipitators, the fans, and in the ductwork

- on the discharge wires of the roaster precipitator

The latter consisting of nodules and rings (donuts).

Dust particles in the effluent after the roaster are zinc oxide. The SO_3 generated in this process cannot exist with the zinc oxide, but forms a zinc sulphate. The rate of this reaction depends upon the penetration of the SO_3 through the zinc sulphate shell towards the zinc oxide core. Thus, the surface of the dust particles consists of zinc sulphate and some "free" SO_3 exists in the gas.

Roaster gas contains a noticeable H_2O component and the "stickiness" of the dust can be plotted as a function of the H_2O component with SO_3 as a parameter. Obviously, "stickiness" is not a physical or technical quantity, thus, no dimensions can be given. The graph (Eisert[9], Figure 12) shows that the stickiness of the dust increases with increasing H_2O content, accelerated by an increasing SO_3 content.

The H_2O content of the roaster gas is caused by the moisture content of the raw material (approximately 8 percent), of the combustion air, and the water sprayed into the roaster for temperature control.

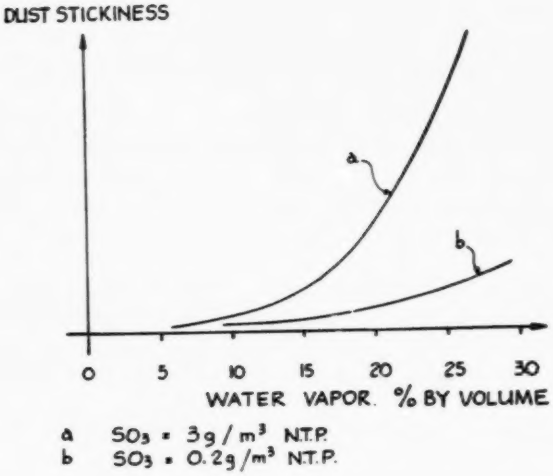

Figure 12: "Dust Stickiness" as a function of water vapor and SO_3 content

Modern roasters produce waste gases with moisture contents of only 8 to 10 percent by volume, and the stickiness of the dust is normally low. A moisture content of 13 percent or more will cause considerable build-ups; and in excess of 20 percent by volume will limit operating periods to not longer than one week at a time. The stickiness of the dust causes undesirable dust build-ups in the waste heat boiler limiting the heat recovery and increasing the gas outlet temperature. Pressure drop in the system increases. Of dire consequences are the dust build-ups in the precipitator causing arcing, and, eventually, short-circuits. The hypothesis of the increase of the stickiness of the dust analog to the increase in moisture content of the gas is based on observations. An exact chemical reaction causing this increase in stickiness has not yet been established, but an analogy to the specific dust resistivity, which is also dependent on the conditions at the gas/dust interface should be mentioned.

The build-ups on the discharge wires (modules) are of a different nature. They are limited to the electrostatic precipitator and can occur with no other build-ups present. They are also zinc sulphates, formed by a different mechanism. Based on a number of observations, it can be concluded that they are caused by inleakage of cold air upstream to the electrostatic precipitator. The mechanism can be explained as follows:

Cold air causes a local temperature drop into the range of the sulfuric acid dew point, forming a sulfuric acid mist. Since the rate of evaporation in this temperature range is rather slow, the sulfuric acid mist reaches the electrical field of the precipitator. Thus, the precipitator collects dust and sulfuric acid mist. The predominant collection of the dust particles is on the collecting surface plates, whereas the sulfuric acid mist particles are predominately precipitated on the discharge wires. Thus, the ratio dust/mist is in favor of dust on the plates and in favor of mist on the discharge wires. The dust can be removed from the plates by normal rapping, but the wires show a layered build-up, which in one case reached about 200mm with an air in-leakage of less than two percent. The build-ups on the wires are relatively sticky and spongy at operating temperature, and, thus, cannot be easily removed by rapping.

In general, plate build-ups can be reduced by reduction of the moisture content of the waste gas, and wire build-ups can be reduced and often eliminated by preventing in-leakage of cold air.

Review of Typical Installations

A typical gas cleaning installation for a roaster may include --

- cyclones
- hot-gas precipitator
- scrubber
- gas cooler
- wet process precipitator

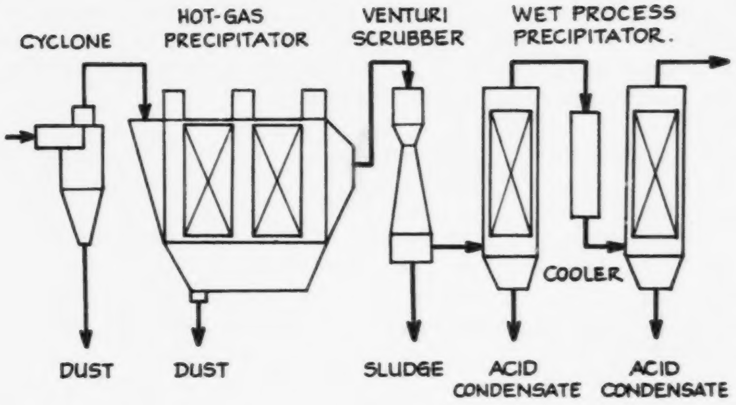
CYCLONE
HOT-GAS PRECIPITATOR
VENTURI SCRUBBER
WET PROCESS PRECIPITATOR.
COOLER
DUST
DUST
SLUDGE
ACID CONDENSATE
ACID CONDENSATE

Figure 13: Gas Cleaning System (Schematic)

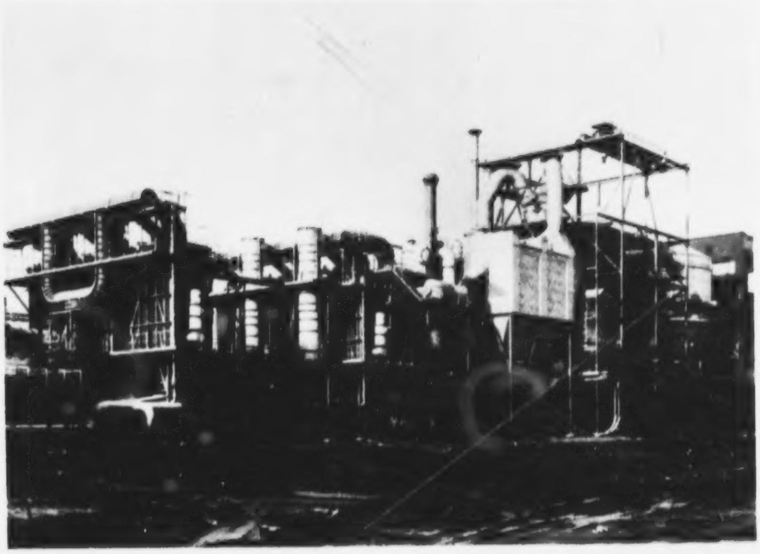

Figure 14. Gas cleaning system for turbulent layer roaster

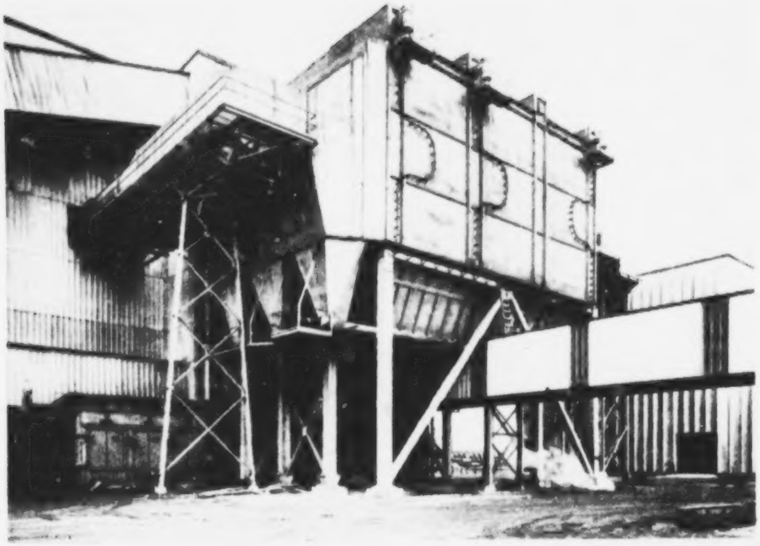

Figure 15. Hot gas precipitator for lead sinter roaster

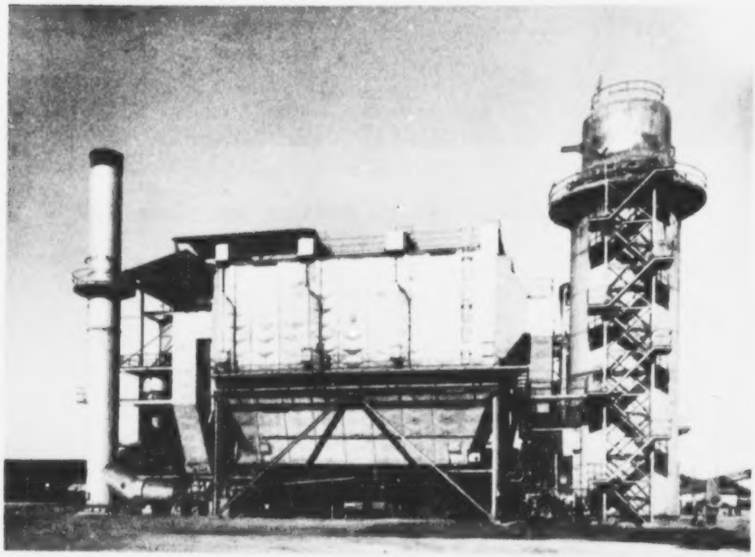

Figure 16. Electrostatic precipitator with gas conditioning tower

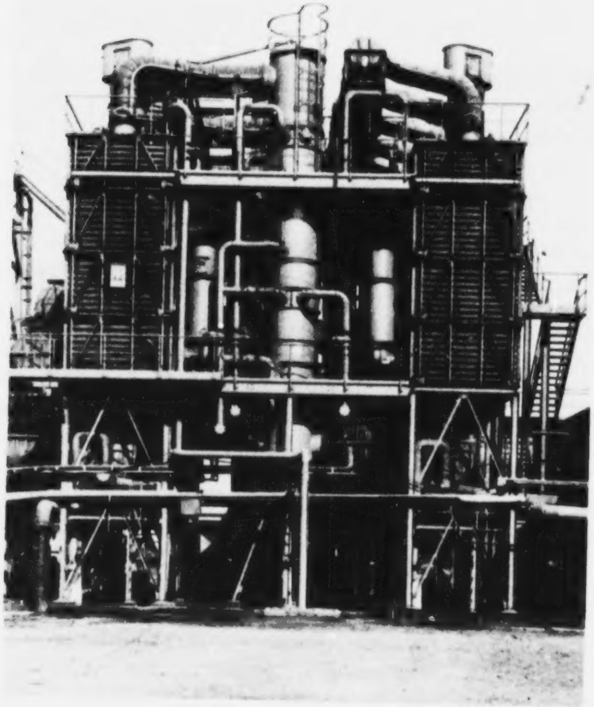

Figure 17. Wet-process lead precipitator with gas cooler

A hot gas precipitator for gas from a sinter machine for Pb/Zn - concentrates for a gas volume of 100,000 Nm^3/hr. at 300 to 400°C. is shown in Figure 15.

Using a conditioning tower allows operation of the precipitator in a more favorable temperature range. An installation to collect lead-oxides is shown in Figure 16. A wet-process precipitator with two electrical fields in series, separated by a gas cooler, is shown on Figure 17. This precipitator and its electrode system are made out of lead. This design is now replaced by precipitators using plastic as the basic material of construction as shown on Figure 18 for an installation with a gas volume of 50,000 Nm^3/hr.

Summary

Electrostatic precipitators are available in a wide range of designs suitable for applications in the nonferrous metals industry. Highly specialized gas cleaning systems have been developed for smelters, including mechanical collectors, dry-and wet-process precipitators, scrubbers, and fabric filters. Although the increasing demands on air pollution control systems will require more use of two-stage systems, i.e., a dry-process collector followed by a wet-process collector, it is expected that electrostatic precipitators will almost always be the main device in each system.

But it has to be recognized that, in some instances, the capital investment necessary to provide adequate performance, i.e., collecting efficiency will be very high and that in other cases, the operation of the gas cleaning system may not be without problems. In those cases, a search for better solutions has to center on the process itself. A typical example is changing the process to operate above the acid dew point to eliminate operational and corrosion problems in the gas cleaning system.

Of equal importance is to avoid dust build-ups in the dry-process precipitator. It is not practical to provide mechanical means of cleaning by rapping for all precipitator internal parts. The magnitude of accelerations to keep, for example, support systems clean of build-ups would be such that these components would require expensive high-strength materials of construction. Thus, it becomes important to reduce or eliminate conditions which could result in dust build-ups. Surface characteristics of particles and their reaction with the surrounding gas have to be influenced to prevent particles from becoming "sticky" by chemical reactions and surface condensation.

To do this, changes in the metallurgical process may again be required. The extent of these changes is still a matter of experience, since reliable data and information on the actual reactions and phases of all materials present in the process are not fully available.

In the past, gas cleaning systems were sometimes regarded as a nuisance for the plant and were often neglected in design, operation, and maintenance. But they have to be considered essential to the operation of the plant and treated accordingly.

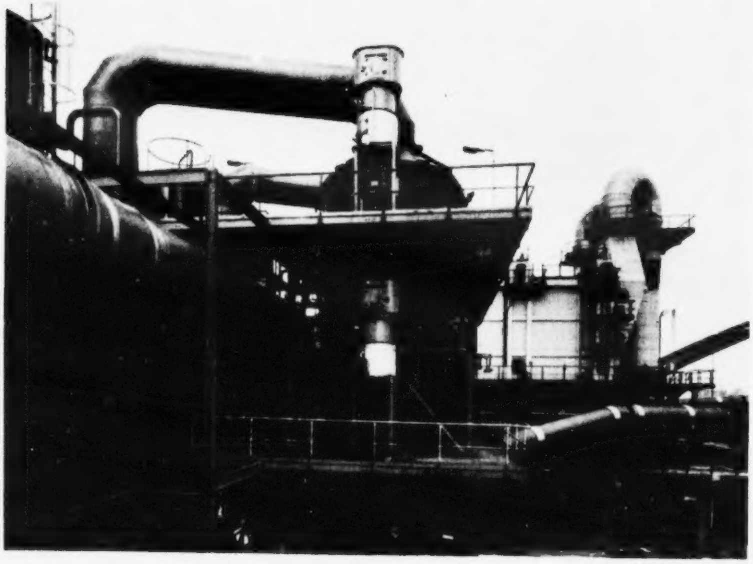

Figure 18. Wet-process plastic precipitator

Acknowledgement:

All photographs courtesy of Lurgi Umwelt und Apparate-Technik Gmbh,
Frankfurt, Germany

During the planning of gas cleaning systems for the nonferrous metals industry, especially when new processes are involved, it has to be considered that the characteristic of the gas/dust system are decisive for the efficiency and operation of the gas cleaning system and that these characteristics result from the metallurgical process and are influenced also by changes upstream of and in the dust collector. An optimum solution of the gas cleaning problem, thus, requires close co-operation between the operator of the plant, his engineer, and the designer of the air pollution control equipment.

ACKNOWLEDGEMENT

The assistance of Dipl. Ing. Werner Eisert of Lurgi Umwelt und Chemotechnik GmbH, Frankfurt, Germany, in the preparation and review of this paper is gratefully acknowledged.

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ALUMINUM INDUSTRY

Session Chairman: Seymour Epstein
The Aluminum Association
Washington, D.C.

AN INTRODUCTION TO THE PRIMARY ALUMINUM INDUSTRY AND ITS COMMONLY EMPLOYED EMISSION CONTROL METHODS

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ABSTRACT

Aluminum bearing ores make up about eight percent of the earth's crust. Aluminum occurs in nature in a variety of forms but the most important naturally occuring aluminum bearing ore is bauxite.

Alumina is extracted from bauxite by the Bayer process. Alumina is then converted to aluminum in primary aluminum plants.

Alumina is electro-chemically reduced in primary aluminum plants to aluminum by the Hall-Heroult process. This process basically involves two steps. First, alumina is dissolved into molten cryolite. Second, an electric current is passed through this mixture, thus decomposing the alumina into metallic aluminum.

The electro-chemical reaction which takes place to produce aluminum is carried out in electrolytic cells or "pots". Aluminum reduction cells are of two general types: Soderberg cells and prebake cells.

In many primary aluminum plants, cell hoods are employed to capture the particulate matter and gases evolved from the reduction cells. These captured gases are, at some prebake plants, directed to an air pollution control system consisting of dry scrubbers. At many Soderberg plants, captured cell gases are directed to wet electrostatic precipitators. Both dry systems and wet electrostatic precipitators have demonstrated a high degree of efficiency in removing particulate matter.

INTRODUCTION

Aluminum is the "magic metal".[1] The following excerpt from a recent article in National Geographic accurately captures the essence of aluminum and provides insight into the ever increasing role which aluminum is playing in the world.

> "It is 1942: A Nazi U-boat surfaces in foggy darkness off a lonely Long Island beach and four invaders land stealthily. Armed with sophisticated explosives, the saboteurs seek to cripple America's burgeoning air armada. Among the targets - aluminum smelters in New York and Tennessee. Their plan - destroy the cables that carry electricity for processing the aluminum. Long before power can be restored, the molten metal will solidify in the furnaces (reduction cells) so that only blasting can remove it, knocking out the smelters for months.
>
> Before the enemy agents can strike, the FBI scoops them up. But the lesson is clear. The light, shiny metal that began its working life as kitchen pots and pans has emerged as a sinew of industrial society." [1]

The Aluminum Association has recently launched a program to make the public more aware of the essential role which aluminum plays in each American's life. The following excerpt points out that aluminum plays an important role in each citizen's day-to-day activities and is vitally important to our nation's efforts to recycle and reuse limited resources and to conserve energy.

> "America needs aluminum for cutting down fuel consumption in transportation. For packaging. For building. For recycling. For transmitting electricity. For protection from the heat and cold. America needs aluminum because it's light weight and strong and versatile. In fact, Americans need aluminum for a thousand different reasons every day of the year."[2]

The remainder of this paper will provide an overview of the primary aluminum industry and the emission control techniques which are commonly employed. By necessity, the paper will be directed towards providing the reader with a general overview of the subject matter.

PRIMARY ALUMINUM CAPACITY

The primary aluminum capacity in the United States, at the end of 1977, was 5,193,350 (short) tons. The worldwide primary aluminum capacity was approximately 15,000,000 (short) tons at the end of 1977. [3] As such, for the year 1977 the primary aluminum industry in the United States accounted for approximately thirty-five percent of the worldwide primary aluminum capacity.

The companies which produce primary aluminum in the United States, and their respective rated capacities, are shown in Figure 1.[3] "The percentage distribution of U.S. primary capacity by company at year-end 1977 was: Alcoa 32.3; Reynolds 18.8; Kaiser 13.9; Consolidated 6.8; Anaconda 5.8; Alumax 4.2; Howmet 4.2; Martin Marietta Aluminum 4.0; Revere 3.9; Noranda 2.7; National Aluminum 1.7; and Southwire 1.7." [3]

Figure 1. PRIMARY ALUMINUM PRODUCTION CAPACITY [3]
AT THE END OF 1977
(Short Tons)

Company	Annual Capacity
Aluminum Company of America	1,675,000
Alumax	218,700
Anaconda Aluminum Company	300,000
Consolidated Aluminum Corporation *	351,950
Martin Marietta Aluminum	210,000
Howmet Corporation	218,700
Kaiser Aluminum & Chemicals Corporation	724,000
National-Aluminum	90,000
Noranda Aluminum Company	140,000
Revere Copper & Brass, Inc.	200,000
Reynolds Metals Company	975,000
Southwire	90,000
	5,193,350

* Includes the aluminum capacity acquired from Olin Corporation in 1974.

GEOGRAPHICAL DISTRIBUTION OF PLANTS

Figure 2 shows the graphical distribution of primary aluminum plants located in the United States.[4] As depicted in Figure 2, there is a concentration of primary aluminum plants in the Pacific Northwest, the Tennessee Valley region, the Ohio Valley region and the Gulf Coast area. The locations of the existing plants were selected after careful consideration of the numerous factors which must be considered in the site selection process; however, the availability of an adequate supply of electrical energy has traditionally been an important factor in site selection decisions.

GENERAL DESCRIPTION OF THE PRIMARY ALUMINUM REDUCTION PROCESS

Basic Cell Construction and Operation

In the Hall-Heroult process, aluminum is produced by the electrolysis of alumina dissolved in a fused salt bath. The reduction of alumina to aluminum is carried out in reduction cells, or, as they are commonly called, pots. A reduction cell is essentially a rectangular steel shell which is lined with heat insulating materials and with carbon blocks. The carbon liner of the cell serves as the cathode for the electrolytic cell. The carbon liner contains collector bars which are connected to a collector bus, which is, in turn, connected to the main potline service bus.

The anode, or anodes, of a reduction cell are also composed of carbon. The anode is suspended over the cell and is immersed in the molten electrolyte bath.

Individual reduction cells are electrically connected in series and are typically housed in buildings, commonly referred to as potlines. A single plant typically has two to as many as seven potlines with up to 1000 cells.

The basic reduction process begins when the cell is fed a supply of cryolite and alumina. Direct electrical current is then passed through the alumina/cryolite solution causing the solution to decompose into metallic aluminum and oxygen. The aluminum migrates to the bottom of the cell where it can later be siphoned off. The oxygen released in the decomposition reaction combines with the available carbon and is released from the cell. The two materials consumed in the reduction process are alumina, which is periodically fed to the cell, and carbon, which is consumed from the anode. A small amount of cryolite is also consumed in the electrolytic production of aluminum. A general material flow diagram for a primary aluminum plant is shown in Figure 3.

The reduction process is carried out in two main types of reduction cells, prebake and Soderberg. The characteristics of each type of cell are described in the sections which follow.

Figure 2. GEOGRAPHICAL DISTRIBUTION OF PRIMARY ALUMINUM PLANTS[4]

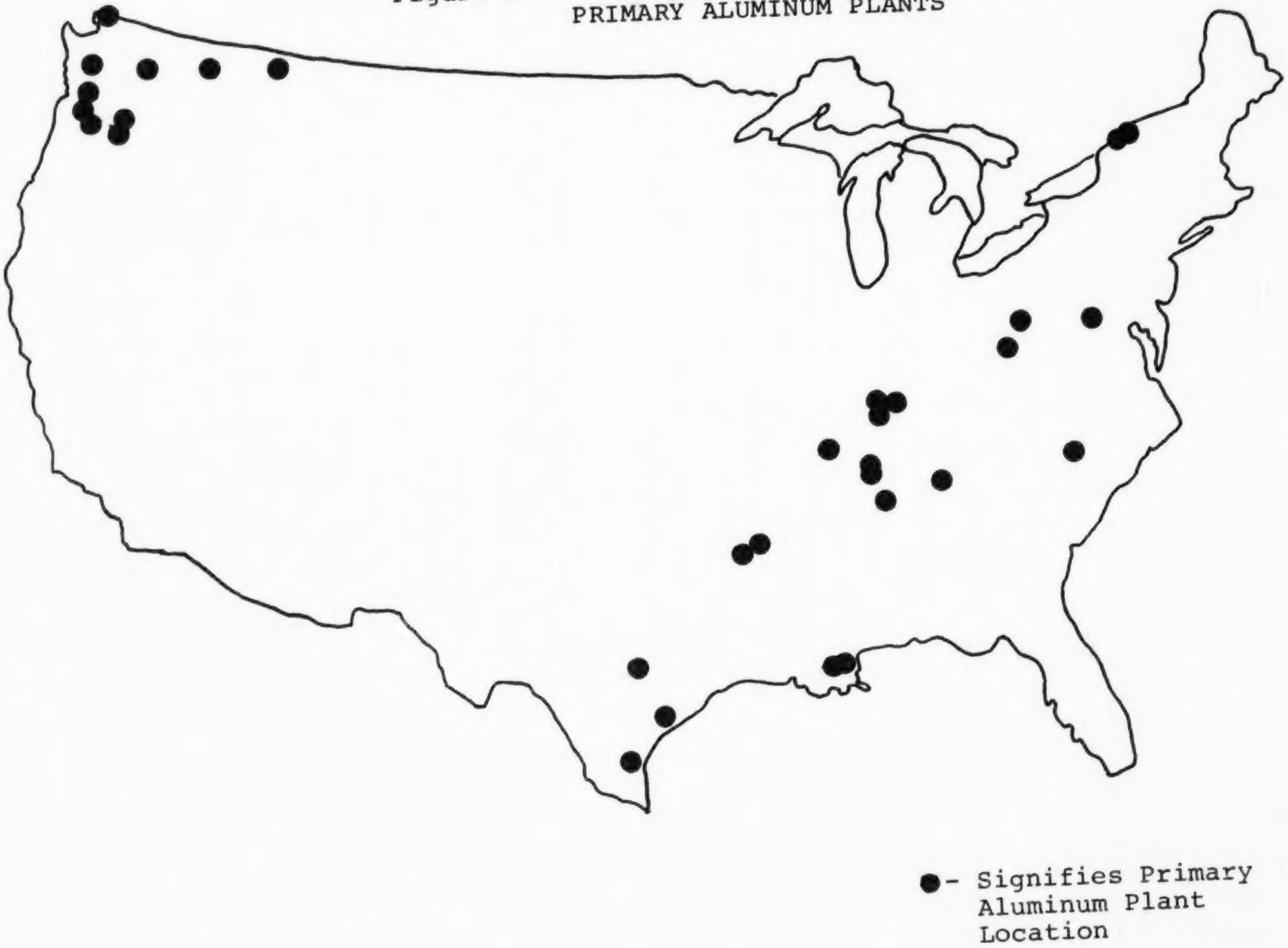
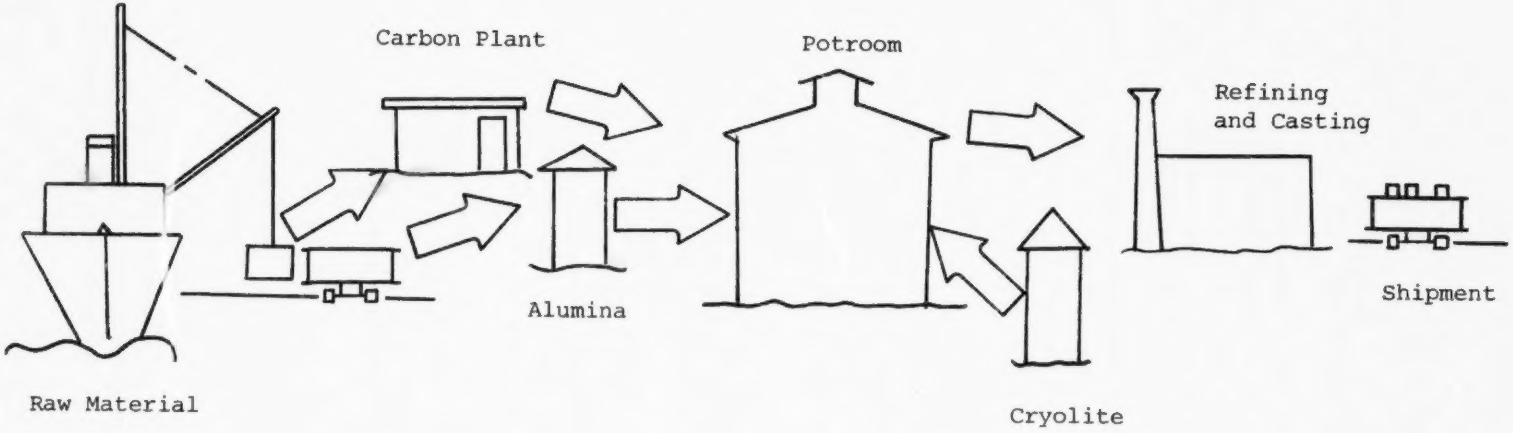

● - Signifies Primary Aluminum Plant Location

Figure 3. MATERIAL FLOW DIAGRAM FOR A PRIMARY ALUMINUM PLANT [6]


Carbon Plant
Potroom
Refining
and Casting
Raw Material
Alumina
Shipment
Cryolite

Prebake Cells

Two basic variations of prebake (PB) cells are employed in the primary aluminum industry. One type is called a center-work prebake cell while the other is called a side-work prebake cell. These two types of prebake cells are illustrated in Figures 4 and 5.[5]

Prebake cells, as their name implies, employ prebaked carbon blocks to serve as the cell anodes. The prebaked anodes are pressed, or jolted, from a mixture of petroleum coke and pitch binder and baked prior to use in the cell. The prebake anodes are progressively lowered into the electrolyte as they are consumed and, of course, eventually replaced by new anodes.

The two variations of the prebake cell which are employed in the primary aluminum industry are, as previously mentioned, center-work and side-work cells. A center-work prebake cell, which is shown in Figure 4, derives its name from the manner in which the cell is "worked". In the case of a center-work prebake cell, alumina is fed to the cell along its centerline and the crust is also broken along the centerline of the cell.

In the case of a side-work prebake cell, the cell is "worked" from the side. A side-work prebake cell is shown in Figure 5.

Soderberg Cells

Soderberg cells are also commonly found throughout the primary aluminum industry. The two types of Soderberg cells used in the industry are the horizontal stud Soderberg (HSS) cell and the vertical stud Soderberg (VSS) cell. These cells are illustrated in Figures 6 and 7.[5]

In both types of Soderberg cells, the carbon anode is contained in a rectangular steel shell suspended over the cell. The anode can be lowered and, as carbon is consumed, carbon paste is added periodically to the top surface of the anode. The heat of the cell progressively bakes the new material as the anode is lowered.

Figure 6 shows the general arrangement of a vertical stud Soderberg (VSS) cell. The steel studs carrying electrical current to the cell are inserted into the anode on a staggered schedule and become firmly baked into position as they approach the cell working surface.

In this type of cell the anode casing typically carries a small hood or skirt which collects the gases evolved around the periphery of the anode. These gases are normally burned in a special type of burner located close to the "gas skirt", as shown in Figure 6.

The horizontal stud Soderberg (HSS) cell is shown in Figure 7. It is similar to the VSS cell, except that the steel studs are inserted through the long sides of the anode casing. Crust breaking operations, and the addition of new alumina, are carried out along the sides of the cell, as is the case with VSS cells.

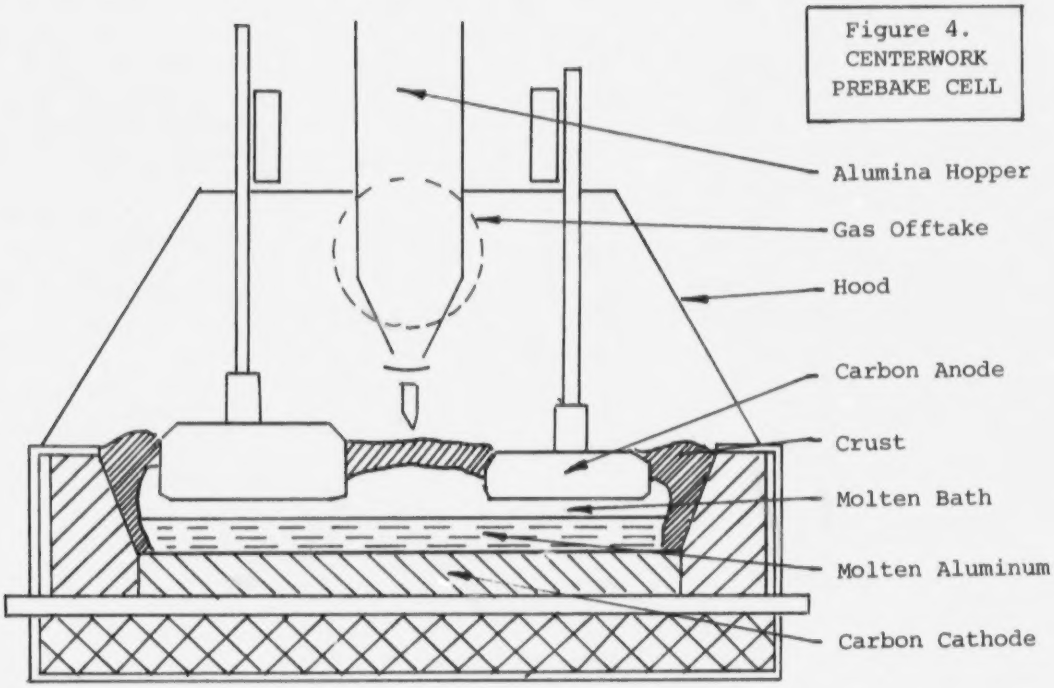


Figure 4.
CENTERWORK
PREBAKE CELL

Alumina Hopper
Gas Offtake
Hood
Carbon Anode
Crust
Molten Bath
Molten Aluminum
Carbon Cathode

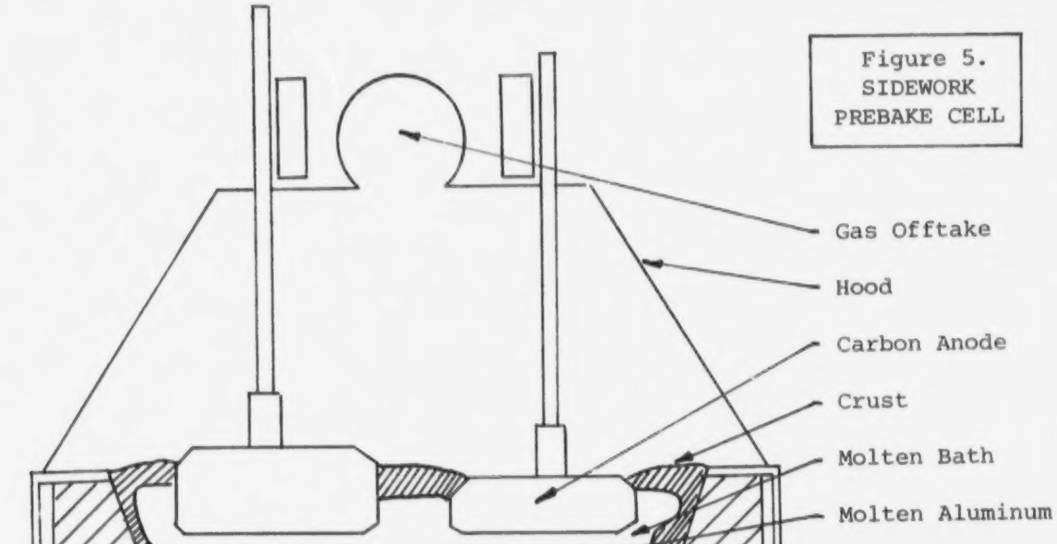


Figure 5.
SIDEWORK
PREBAKE CELL

Gas Offtake
Hood
Carbon Anode
Crust
Molten Bath
Molten Aluminum
Carbon Cathode

Figure 6.
VERTICAL STUD
SODERBERG CELL

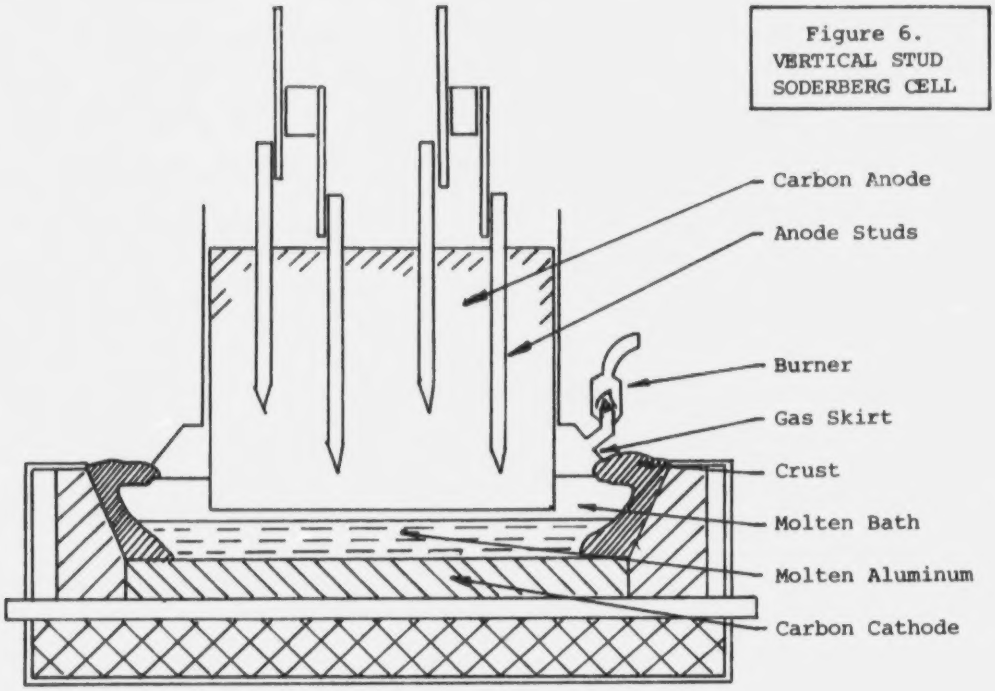
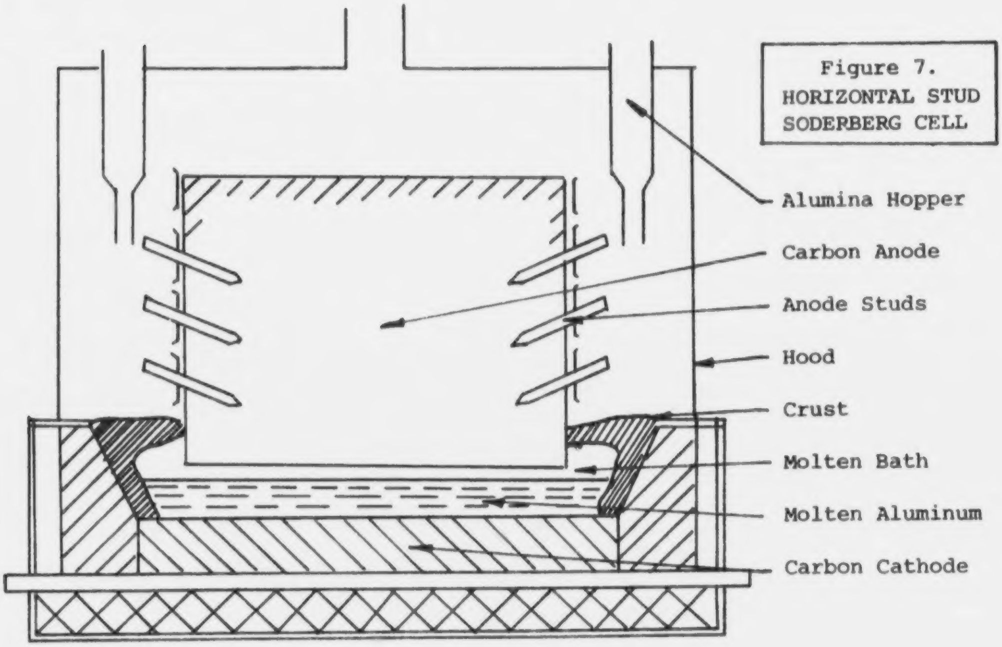

Carbon Anode
Anode Studs
Burner
Gas Skirt
Crust
Molten Bath
Molten Aluminum
Carbon Cathode

Figure 7.
HORIZONTAL STUD
SODERBERG CELL


Alumina Hopper
Carbon Anode
Anode Studs
Hood
Crust
Molten Bath
Molten Aluminum
Carbon Cathode

EMISSION CONTROL IN THE PRIMARY ALUMINUM INDUSTRY

Primary Fume Collection

In the primary aluminum process, certain gaseous and solid constituents are evolved from the reduction cells. The amount and type of evolved constituents is a function of a number of operational conditions such as: bath ratio, bath temperature, bath chemistry, and the frequency of anode effects.

The major constituents evolved from the reduction cells are particulate matter along with gaseous and particulate fluoride. The exact amounts of each of the constituents evolved will, as previously stated, vary with a number of operational conditions. In fact, the evolution rate would be expected to vary, for a variety of reasons, even within an individual plant employing uniform operational conditions.

Efficient cell hoods can be employed, in cases where they are necessary, to capture a high percentage of the constituents evolved from the individual reduction cells.[*] The small portion of the evolved constituents which escape capture by the primary collection system are released through the potroom roof ventilators. The control of these secondary emissions is not routinely practiced in view of the low concentration of constituents found in the large volume of air exhausted from the potroom roof ventilators. Further, control of secondary emissions is not routinely practiced since it is not cost or energy effective. Control of the dilute, large volume secondary air stream requires a large energy and financial investment to achieve a relatively small amount of additional emission reduction.[+]

The basic purpose of the primary collection hoods is, of course, to capture the constituents evolved from the cells and direct them to the primary emission control system. A variety of successful hood designs have been employed within the primary aluminum industry. Typically, hood design efforts require a considerable amount of engineering and research time in order to tailor the hood design to each individual plant's physical facilities and

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[*] It should be noted that cell hoods are not necessary or warranted in some individual situations. Fume collection for VSS cells is, in many ways, dissimilar to fume collection for other types of cells. For this reason, fume collection for VSS cells will not be addressed in detail in this paper.

[+] The conclusion as to the general infeasibility of secondary emission control is supported by EPA's proposed position on emission guidelines for existing primary aluminum plants. In the proposed guidelines, EPA concludes that secondary control for HSS and CWPB "does not appear justified" and that secondary control at VSS and SWPB plants is only justified depending "on (the) severity of the fluoride problem," ("Draft Guideline Document: Control of Fluoride Emissions from Existing Primary Aluminum Plants," U.S. Environmental Protection Agency, June 1978, Table 1-4).

Primary Emission Control Equipment

Over the years, a variety of air pollution control equipment has been successfully employed in the primary aluminum industry. In recent years, the most common control techniques employed in the primary aluminum industry have involved the installation of spray towers, wet electrostatic precipitators (WESP's) and dry scrubbing systems. It should be noted, however, that these types of control systems may not be appropriate or necessary at all plants.*

Spray Towers -

Spray towers have long been a traditional method of control in the primary aluminum industry. A number of plants currently successfully employ spray towers to reduce atmospheric emissions. "The term spray tower is applied to gas scrubbing devices in which the gas passes through an enclosure at relatively low velocity and is contacted by water, alkaline liquor or limed water liquor, sprayed from headers usually in counterflow with the gas . . . A typical spray tower . . . uses water or limed water and consists of an open top redwood tower . . . with cyclonic inlet breeching and a mist eliminator at the top. Liquor may be sprayed down from the top or at several elevations in the tower."[4]

Spray towers have been found to achieve excellent reductions, in the range of 93 to 99 percent, in the gaseous fluoride level of the captured gases. Singmaster and Breyer have reported that particulate removals of 64 to 80 percent can be achieved by spray towers at primary aluminum plants.+

Wet Electrostatic Precipitators -

The gases captured by the cell hoods can be directed to wet electrostatic precipitators (WESP's). "The electrostatic precipitator is a relatively large chamber through which cell gas streams pass at low velocity, usually 3 to 5 feet per second (ft/sec). In its usual form, high negative voltage corona discharge wires are suspended across the air stream and grounded collector plates form parallel passageways for the air. The ionizing field surrounding the discharge wires ionizes part of the gas stream and imparts electric charge to most particles, some positive but most negative. Positively charged particles migrate toward the discharge wires and negatively charged particles migrate to the grounded collection plates."[6] Water sprays are then typically employed to wash the collected materials from the plates.

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* In some specific plant locations the high level of removal provided by WESP's, dry scrubbers, or other equipment is not warranted or necessary. A number of factors must be considered in developing a control strategy for an individual plant. Among the factors which must be considered are local conditions, economics, dispersion characteristics, plant operational practices, courtyard space limitations, equipment maintenance considerations, etc.

+ Singmaster and Breyer caution that care should be exercised in applying the reported removal efficiencies for spray towers since the data base for their conclusions was sparse and because of limitations in the collected data.

Wet electrostatic precipitators typically have a scrubbing section in the inlet section of the unit. The water sprays, among other things, help to condition the gas stream. Gas conditioning may be very important since some experts believe that improved removal is achieved, for certain types of particulate, by raising the moisture content of the gas stream.[6] Gaseous fluoride removal is, of course, also enhanced by the water sprays.

Singmaster and Breyer state that wet electrostatic precipitators ". . . operating on potline primary effluent streams report design and operating removal efficiencies ranging from 60 percent to 99 percent on particulate. Unlike many types of control equipment, electrostatic precipitators may be designed for almost any selected efficiency. By using conservative design dimensions, by controlling (the) humidity of the incoming gas, and by operating at high voltage. . . (wet electrostatic precipitators) can achieve 98 to 99 percent removal of potline effluent particulates."[4]

In its deliberations on establishing Section 111d guidelines for existing primary aluminum plants, EPA has proposed that ". . . spray tower-electrostatic precipitator combinations can readily achieve 98.5 percent total fluoride removal."[6] Further, it should be noted that EPA is presently considering establishing 95 percent as the target removal efficiency for wet electrostatic precipitators alone. *

Dry Scrubbing Systems[4]

Dry scrubbing systems have also been successfully applied at a number of primary aluminum plants. Dry scrubbing systems typically employ alumina to absorb gaseous fluoride along with mechanical separation to remove particulate matter. The three types of dry scrubbing systems employed in the industry are: coated filter dry scrubbing system, fluid bed dry scrubbing system, and injected alumina dry scrubbing system. The names of the individual types of dry scrubbing systems are derived from their basic method of operation.

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* At some existing plants which have recently installed new WESP installations, space has not been available to leave existing spray towers. EPA has, however, stated that the level of control corresponding to an installation of spray towers-electrostatic precipitators is not mandatory except in special situations where it may be warranted because of local conditions. This policy was announced by EPA technical staff at the 22 August 1978 meeting of the National Air Pollution Control Techniques Advisory Committee.

In all cases, the alumina used in the dry scrubbing system is eventually returned to the reduction cells. This recycling of the reacted alumina allows the recovery of the major portion of the fluoride value of the captured gas stream without, in most cases, the need for elaborate further processing.

In the coated filter dry scrubbing system, alumina is injected into the gas stream to form an alumina layer on the fabric filter bags. The alumina layer is then available to absorb gaseous fluoride as the gas stream migrates through the alumina layer. The alumina layer also traps a large portion of the particulate matter in the gas stream.

The fluidized bed dry scrubbing systems, as their name implies, employ a fluidized bed of alumina. In this type of system alumina is continuously directed to a fluidized bed reactor and the reacted bed material is allowed to overflow the system. The fluidized bed traps a large amount of the particulate matter in the gas stream and, as previously noted, the alumina absorbs the gaseous fluoride. A baghouse is located above the reactor bed to remove the alumina and trapped particulate matter from the gas stream.

Injected alumina dry scrubbing systems have also been successfully utilized in the primary aluminum industry. In this type of system, alumina is injected into the gas stream and provided with an adequate opportunity for contact with the gas stream. The alumina laden gas stream is then directed to bag filters.

Singmaster and Breyer report that dry scrubbing systems are capable of achieving from 97 to 99 percent removal of particulate matter.[4] Gaseous fluoride removal efficiencies of up to 99 percent have been reported for dry scrubbing systems.[6]

In its deliberations on establishing Section 111d guidelines for existing primary aluminum plants, EPA has proposed that dry scrubbing systems can achieve 98.5 percent total fluoride removal.[6] Further, it should be noted that EPA is presently reconsidering the issue and will likely establish the control efficiency range for dry scrubbing systems at 95 to 98.5 percent.*

FUTURE PERSPECTIVES ON ALUMINUM

The future for aluminum is bright and aluminum justly deserves being called the "magic metal". "Indeed, just as earlier ages of human development have taken their names from the distinctive material that nurtured them - Stone, Bronze, Iron - there are those who believe our era may be called the Aluminum Age." [1] Aluminum ". . . offers a shining hope for energy conservation. Once made, the metal can be recycled over and over for only a fraction of the energy used in making it originally. Also, by putting our overweight autos on a strict aluminum diet, we can drastically reduce their weight and thus their thirst for gasoline " [1]

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* This policy was announced by EPA technical staff at the 22 August 1978 meeting of the National Air Pollution Control Techniques Advisory Committee.

An interesting perspective on the future course of society and the role which aluminum will play was outlined in the recent National Geographic article. Glenn T. Seaborg, former chairman of the Atomic Energy Commission and a Nobel Prize-winning chemist is quoted to say that he ". . . envisions a society in which 'the present materials situation is literally reversed; all waste and scrap - what are now called secondary materials - become our major resources, and our natural, untapped resources become our back-up supplies.' . . .When and if this day comes, versatile aluminum will be playing a role - or maybe two or three."[1]

FUTURE PERSPECTIVES ON ENVIRONMENTAL CONTROL

In the environmental area, the primary aluminum industry can be proud of its recent history. The industry has, where necessary, developed innovative approaches to controlling emissions. The development of functional hooding systems and the successful application of wet electrostatic precipitators and dry scrubbing systems are significant achievements for which the industry can be justifiably proud. The challenges facing the primary aluminum industry in the future will likely center around continuing to optimize the operation of environmental control facilities and their relationship to the process. Further, attention will likely continue to be directed toward the challenges of additional recycling.

ACKNOWLEDGEMENTS

The able assistance of Ms. Janet Ashman in the preparation of this manuscript is greatly appreciated.

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PARTICULATE EMISSION CONTROL IN AN AUTOMATED SIDE-WORK PRE-BAKE ALUMINUM REDUCTION FACILITY

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ABSTRACT

Automated and advanced technology process equipment in an aluminum reduction potline can require added maintenance and supervisory control to continuously assure maximum particulate collection. At the Eastalco Aluminum Company reduction plant at Frederick, Maryland, alumina feeding and crust breaking is accomplished with an internally programmed computer operated pot working machine. The computer opens the pot hoods when working the pot is required and closes the pot hoods on completion of the work. Slight mechanical adjustment changes during the day of either the pot working machine or hoods can result in damage to the hoods by the pot working machines. Such damage, if not corrected, could result in loss of negative draft under the hoods and loss of particulate to the less efficient secondary scrubbers. Manned pot tending machines utilized to change anodes, tap metal and add or remove electrolyte, also require close operator and supervisory attention due to close tolerances around the pot hoods. Eastalco employs a maintenance crew to respond to required adjustment changes and to repair the pot hoods due to equipment malfunction or operator error.

INTRODUCTION

Eastalco Aluminum Company operates a two potline primary aluminum reduction facility south of Frederick, Maryland, with an annual capacity of 176,000 tons. The plant began operation as a one potline facility in 1970. The second potline was constructed over the 1974-1975 period and became fully operational in 1976. The Hall-Heroult reduction process is utilized to produce primary aluminum in 480 cells or pots.

The original one potline configuration utilized Pechiney designed side-work pre-bake pots.[1] Individual pots were not hooded. Control of emissions utilized the potroom as a containment device with pot emissions being collected in sixty-four (64) spray screen wet scrubbers located in the roof of the potroom buildings. Effluent gas flow was maintained by exhaust fans in each roof scrubber producing a negative pressure in the potrooms.[2]

Beginning in 1973, the original potline was retrofitted with hoods over the individual pots. The hooded effluent was ducted to injected alumina (Al_2O_3) dry scrubbers to provide a primary control system. Twelve (12) of the roof wet scrubbers were removed to provide the required room for primary system ducts. The remaining wet roof scrubbers were retained as a secondary control system.[3] The second potline was constructed utilizing side-work pre-bake pots and the identical primary and secondary control systems retrofitted to the original potline.

The retrofit reduced particulate emissions from the original potline by sixty three (63)%, and the entire two potline complex had twenty seven (27)% less particulate emissions than the original one potline complex.

Figure 1 shows an aerial view of the present reduction facility. The four 1500 feet long buildings contain the two potlines. The four injected alumina dry scrubbers can be seen standing between the potline buildings. The stacks of the many secondary control system wet scrubbers can be noted along the roof of each of the potline buildings. Figure 2 illustrates the dual Eastalco emission control system.

Particulate emissions from pre-bake primary aluminum potlines are well documented[4,5] and consist primarily of alumina (Al_2O_3). Other components include carbon (C), cryolite (Na_3AlF_6), aluminum fluoride (AlF_3), calcium fluoride (CaF_2), chiolite ($Na_5Al_3F_{14}$) and iron oxide (Fe_2O_3).

Installation of best achievable control technology at Eastalco obviously had a marked effect on reduction of particulate emissions. As is usually the case, further reductions are achievable only in small increments and require additional demands on both personnel and equipment. Emission control, maintenance and potline personnel at Eastalco are showing success at achieving those increments of progress.

SIDE-WORK PRE-BAKE REDUCTION CELL HOODS

As the development of primary aluminum reduction cells or pots progressed since the Hall-Heroult discovery, a divergence in the pre-bake technology occurred between the United States and Europe. In the United States, alumina feeding and metal tapping was accomplished down the center of the pot and the anodes were widely separated to provide for these activities. Only anode changing required access to the side of the pot. In Europe, the pre-bake pot was designed so that all activities occurred on the sides of the pot and the anodes were placed closer together on the superstructure. Automated pot working machines and manually operated pot tending machines were developed to accomplish the many pot activities along the sides of the pot. The United States pre-bake pot was termed the "center work pre-bake" pot and the European pre-bake pot became known as the "side-work pre-bake" pot.[6]

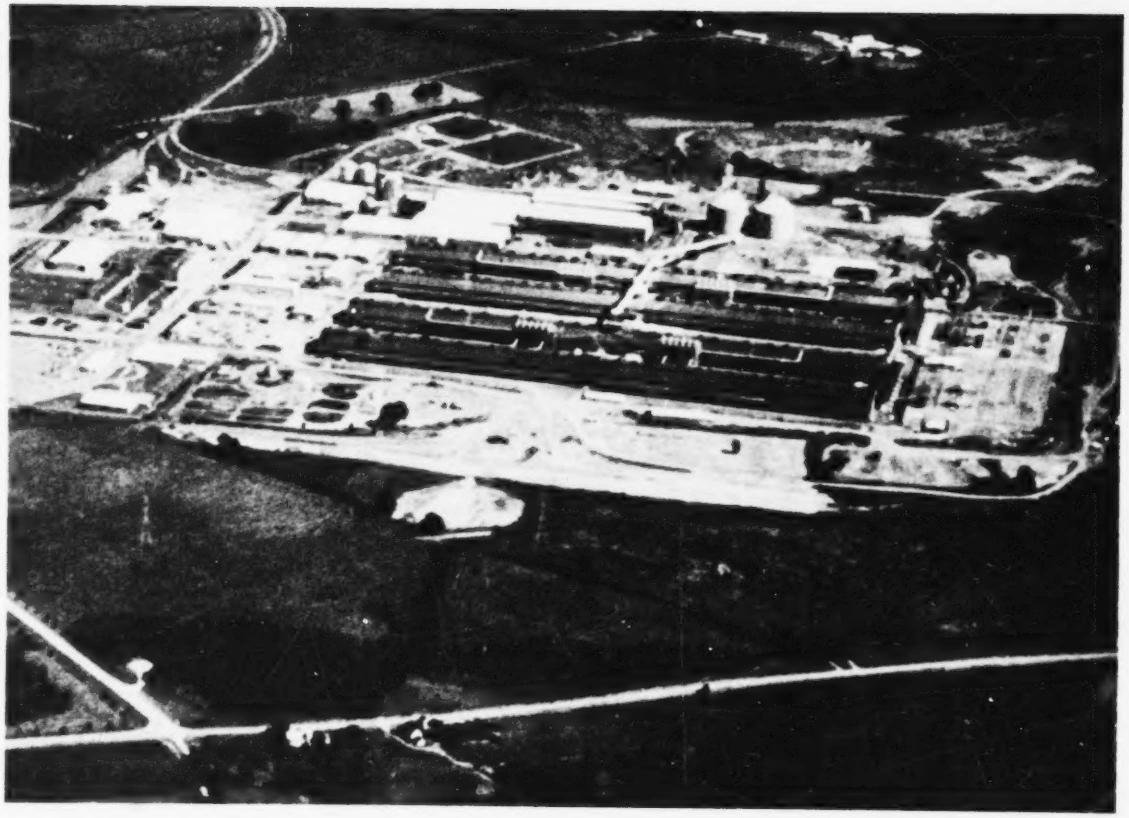

FIGURE 1. Aerial view of Eastalco primary aluminum reduction facility.

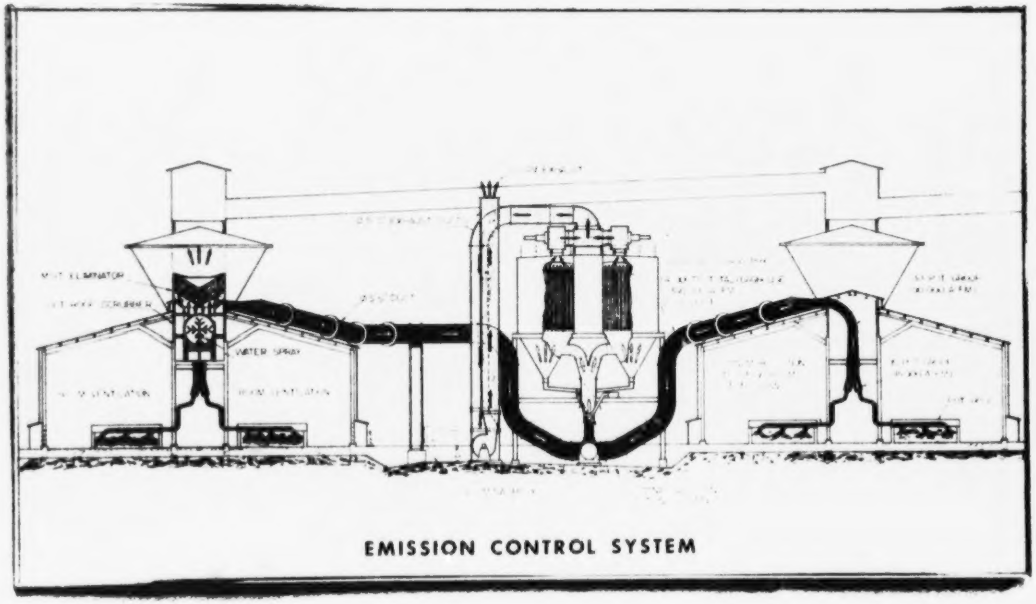
MIST ELIMINATOR
WATER SPRAY
ROOM VENTILATION
EMISSION CONTROL SYSTEM

FIGURE 2. Eastalco Primary and Secondary Emission Control System.

The center work pre-bake pot was very amenable to hooding which provided a relatively low volume high pollutant concentration primary collection stream allowing treatment in classical control systems. Crust breaking and alumina feeding mechanisms utilizable during hood closure, metal tapping through a small end door and removal of only a portion of the segmented hood during anode change were added features in the center work pre-bake pot.

Side-work pre-bake pots, on the other hand, were difficult to hood due to the physical nature of the superstructure and frequent access requirements to the length of both sides of the pot by the pot working and tending machines. The earlier approaches to pollutant control were to use a higher than normal ventilation volume and to treat the ventilation air in roof-mounted wet scrubbers. The entire potroom thus became a hood as such for the pots contained therein.

Beginning in 1970, an extensive effort began to develop a hood for the side-work pre-bake pot. Approximately a dozen different concepts were tested before an acceptable hood was achieved. This allowed hooding of the Eastalco side-work pre-bake pots and installation of a primary control system. This primary control system together with the retained secondary control system then provided a total control system equivalent to the more efficiently hooded center-work pre-bake pot with a primary system control alone.

The features of the hood include a permanent top pan mounted to the superstructure beneath the anode bus. The pan is slotted to allow anode stem to bus contact. Stem seals are mounted on the anode stems to seal off the slotted areas of the pan and, at the same time, to allow stem movement during jacking operations. Movable side doors operated by electrical motors and associated gear boxes provide access to the pot when required by pot working and tending machines. Permanent end panels and wing panels on each end of the pot complete the hood closure. Effluent fume removal is via ducts mounted to the top pan. Figure 3 shows a closed hooded pot. Figure 4 depicts an Eastalco potroom with one side door open on the pot during metal tapping with a pot tending machine. The ducts carrying the pot effluent to the dry scrubbers can be noted between the pot and the wall on the right.

POT HOODS AND AUTOMATED POT WORKING MACHINES

Automated pot working machines termed "semi-gantries" conduct the function of alumina feeding and crust breaking at the Eastalco plant. The semi-gantry conducts its functions on both sides of the pot at the same time and is monitored and controlled by the potline process computer system. The computer has the capability of conducting the semi-gantry activities in various ways, depending upon how the computer is programmed and also sends the semi-gantry to centrally located stations to re-fill its self-contained ore bins when required.

Installation of hoods on the Eastalco pots required modifications to both the pot superstructure and the semi-gantries. The entire superstructure was raised to reduce the potential of top pan damage by the crust breaker during its upward stroke. Since the semi-gantry breaks and feeds both sides of the

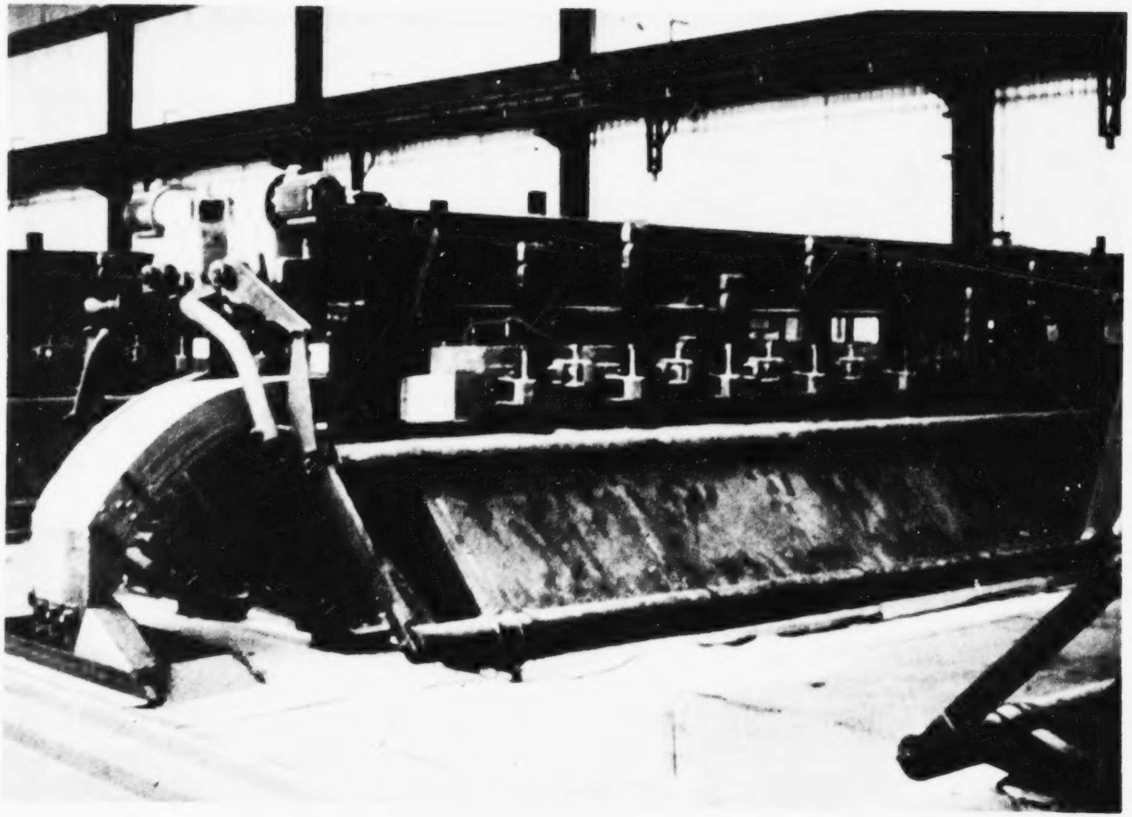

FIGURE 3. Side-work pre-bake hooded pot.

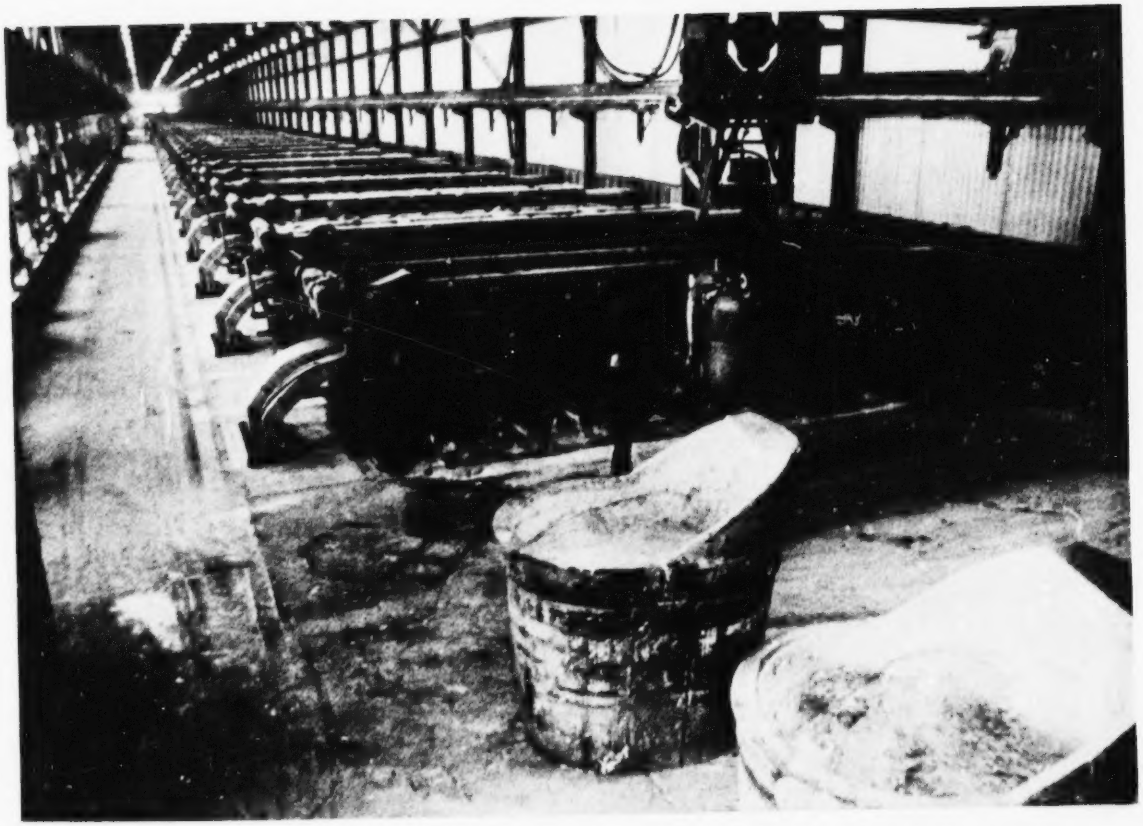

FIGURE 4. Eastalco potroom with one hood door open for tapping operation.

pot at the same time, both hood side doors are required to be in a semi-open position. The bridge on which the semi-gantry travels was raised to provide room between the bridge and the superstructure to accommodate both hood doors in this semi-open position. Hood door opening and closure for semi-gantry operation is also controlled by the potline process computer. Unless both hood doors are in their correct open position, the computer does not allow semi-gantry operation along the sides of the pot. Figure 5 shows the semi-gantry breaking the crust with the hood doors in the semi-open position.

With the hood retrofit, an area with very close tolerances exists between the moving semi-gantry and the tubular arms on the hood side doors as they are raised by the computer to facilitate semi-gantry operation. Figure 6 depicts this very close tolerance. If the semi-gantry trolley hits the tubular arm nut, the tubular arm can be bent which causes binding of the door motion. This causes further strain or warping of the pivot points and gear box mechanisms and therefore improper door closure. The nut can be completely sheared off and cause the door to fall down on that end of the pot. Limit switches on the semi-gantry do not allow motion of the semi-gantry if the trolley moves forward toward the pot. Setting of these limit switches is sometimes difficult due to wear of the trolley rails, which can cause the trolley to drift forward. The weight of the cables carrying electrical service to the semi-gantry along the bridge also has enough weight to pull the trolley forward if they are not properly supported.

Other factors influencing hood damage during semi-gantry operation include damage to the top pan from the crust breaker if the stroke of the crust breaker is out of proper adjustment. Likewise, if the clutches slip on the hood door mechanism, the doors can fall on the semi-gantry while it is working the pot. This results in damage, including loss of alignment of the hood door.

Hood damage, if allowed to occur and to routinely continue to occur, can result in a loss of negative draft under the hoods and therefore reduced pick-up of the particulate effluent by the highly efficient primary control system, resulting in collection of a portion of the particulate by the less efficient secondary control system. This problem is magnified if repairs to the hood by the hood repair crew require portions of the hood to be removed from the pot.

As the above noted difficulties have manifested themselves since the hood retrofit, additional supervisory and maintenance efforts have been initiated to inspect and make the required repairs and adjustments on a daily basis. These repairs and adjustments can include replacement of worn trolley rails, proper setting of limit switches, and adjustment of cable supports, crust breaker stroke and clutches on the hood door mechanisms. These efforts have resulted in a significant reduction in hood damage and they are playing an important role in the achievement of reduced particulate emissions from the Eastalco potlines since the hood installation.

POT HOODS AND MANNED POT TENDING MACHINES

Pot operations such as anode changing, aluminum metal tapping and transfer of liquid electrolyte at Eastalco utilize manned pot tending machines. These manned pot tending machines can also perform the semi-gantry functions.

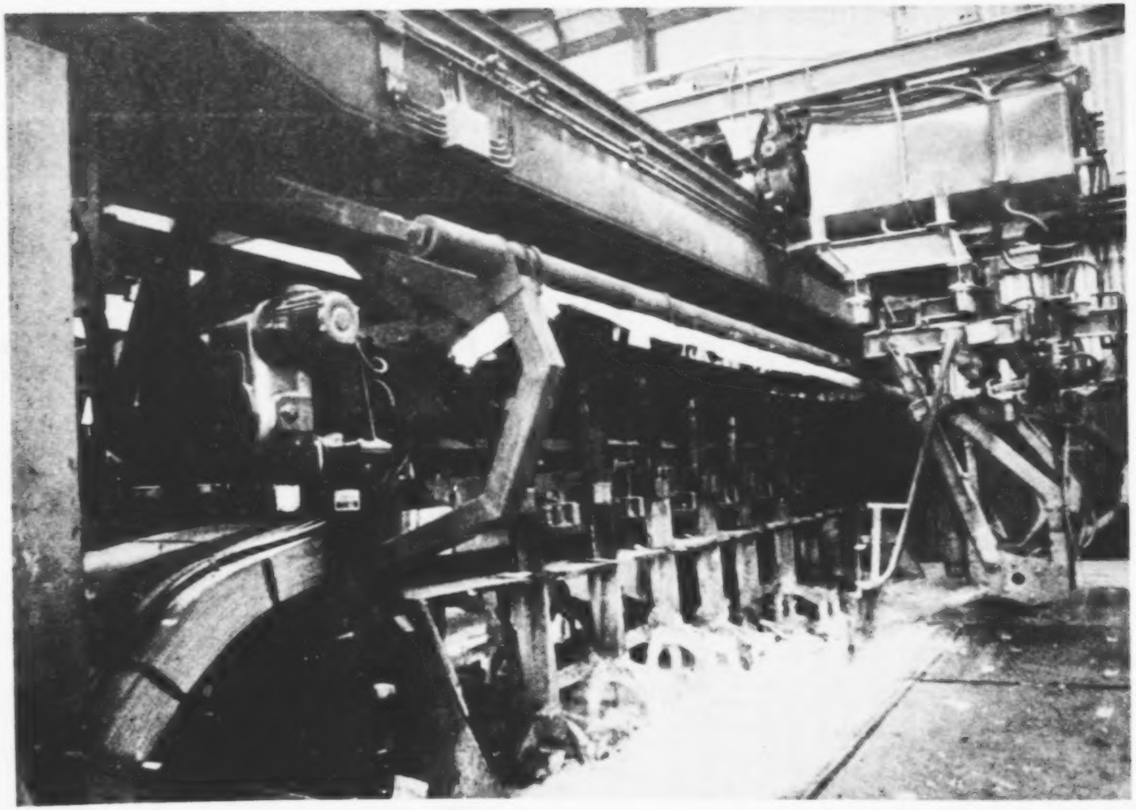

FIGURE 5. Semi-gantry breaking crust with hood doors in semi-open position.

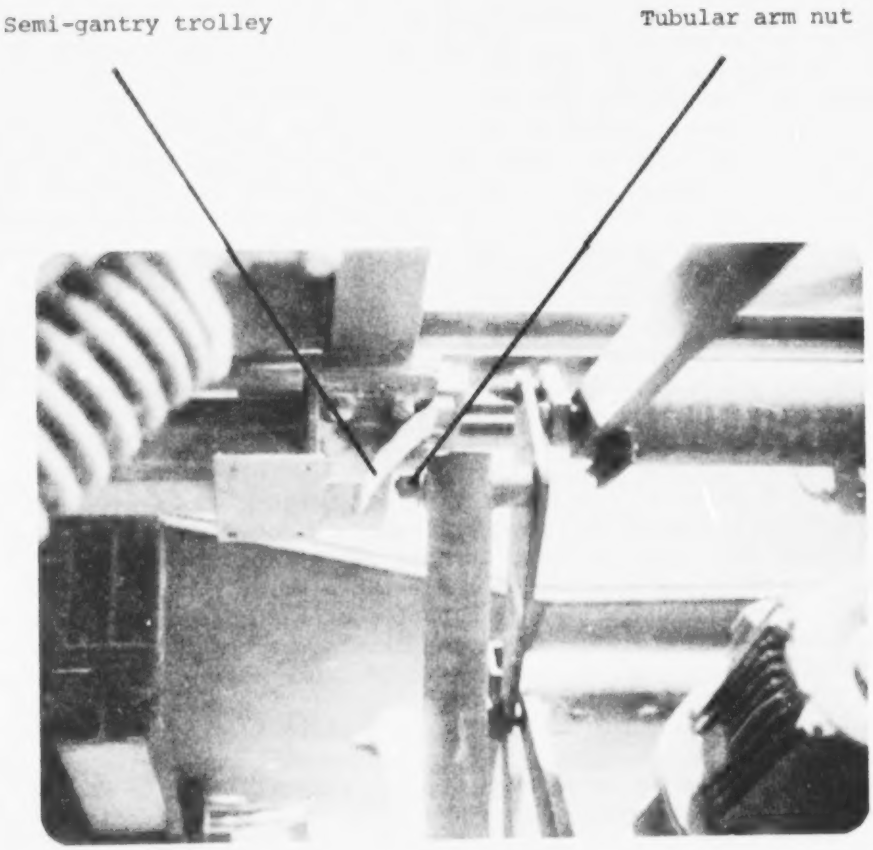
Semi-gantry trolley
Tubular arm nut

FIGURE 6. Close tolerance between semi-gantry trolley and hood tubular arm nut.

Due to close tolerances around the hoods, close operator attention is continuously required to prevent damage to hood components. A responsibility of the potroom supervisor is to assure that the operator is adequately trained to perform the required functions correctly and understands the consequences of hood damage.

During anode change, the fixed top pan can be struck by the anode assembly if excess crust is not swept out of the way prior to moving the assembly into position. Moving the anode stem into the assigned slot in the pan also requires operator diligence to assure that the pan is not bent by the stem. Figure 7 shows an anode assembly being moved into place. Similar problems occur while tapping metal or transferring liquid electrolyte. The tap hole must be located correctly and be large enough so the spout does not hang up during insertion or removal. The tapping spout can strike the pan and bend it or break the pan's angle supports and the pan will then sag. Figure 8 shows the proper tapping procedure.

The wing panels on the ends of the pot can also be damaged during changing of end anodes. Operator care and attention are required to assure that such anodes are set in without striking these wing panels and requiring repair or replacement by the hood repair crew.

During anode change, metal tapping and liquid electrolyte transfer, the hood side door is opened on only one side of the pot. However, to accomplish these required activities, the hood side door is moved to its full open position. This full open position is best illustrated in Figure 4. During anode changes, it is important that the crane also be moved inward with extreme care. If the crane strikes the bottom of the hood side door during its inward movement, the door is pushed across the top of the superstructure. This can result in considerable damage to both the door and the arm supports.

Considerable accomplishment has been achieved in supervisor training to assure operator care when using pot tending machines around the retrofitted pot hoods. Operator experience over time is also a factor in reducing hood damage. The problems discussed herein are becoming the exception rather than the rule. As was the case with the automated pot working machines, these improvements are reducing particulate emissions from the Eastalco potlines.

SUMMARY

The installation of hoods and ducting of the hoods to injected alumina dry scrubbers to provide a primary control system and the retention of the existing wet roof scrubbers as a secondary control system reduced particulate emissions from Eastalco's original side-work pre-bake potline by 63 %. Construction of the second potline identical to the environmentally retrofitted first potline resulted in 27 % less particulate emissions from the two potline complex than was the case with the one potline complex.

Further reductions have been achieved in small increments. Figure 9 illustrates the reduction in grain loading from the secondary wet scrubbers

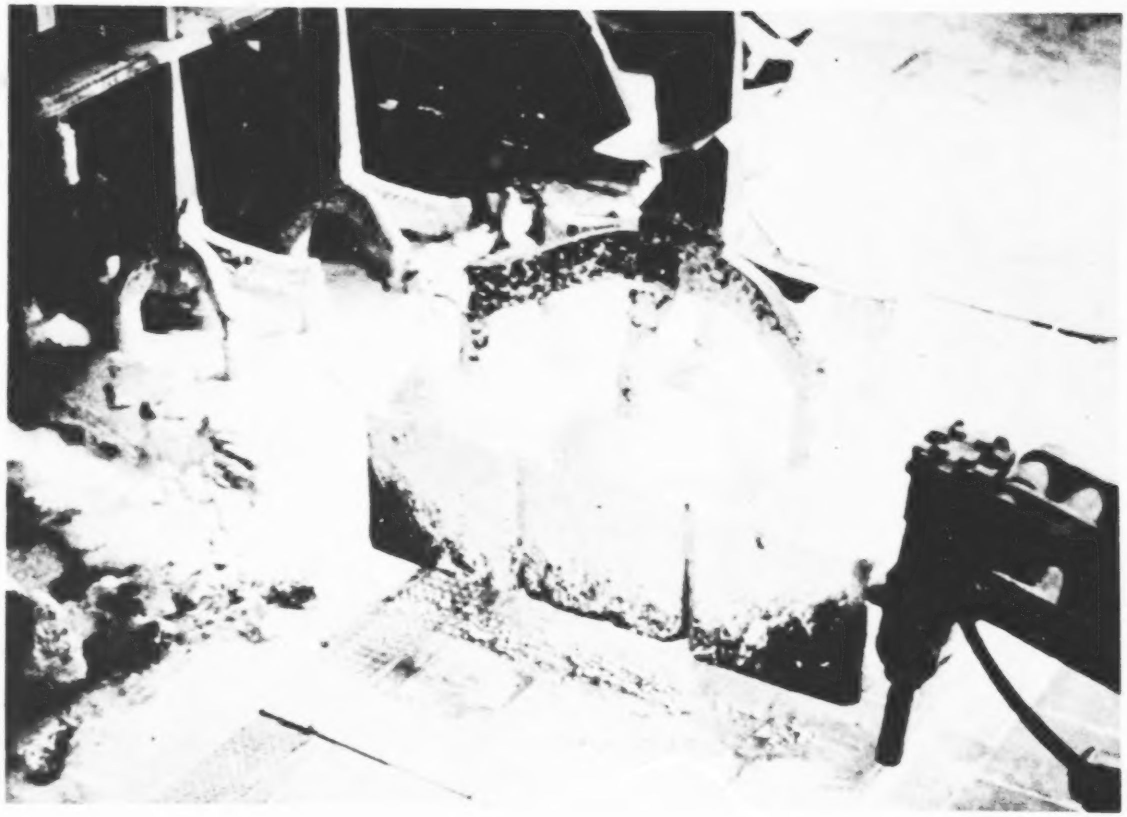

FIGURE 7. Anode assembly being moved into place on the pot.

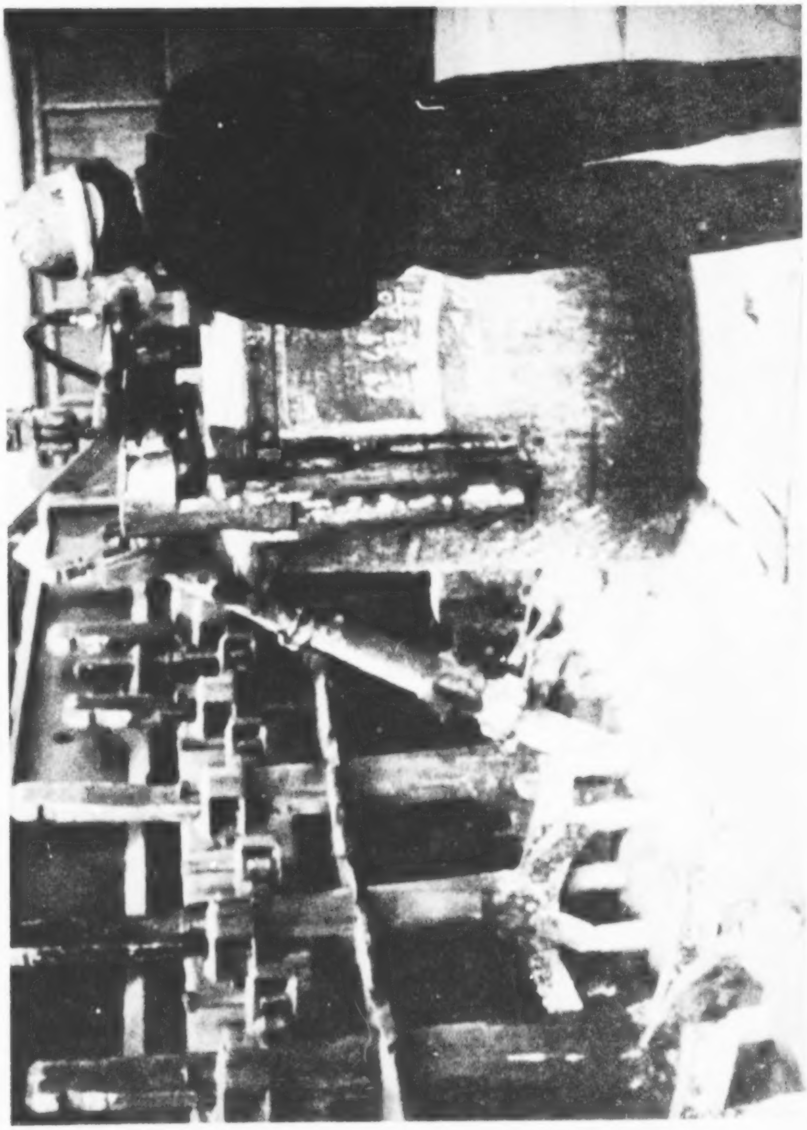

FIGURE 8. Aluminum metal tapping.

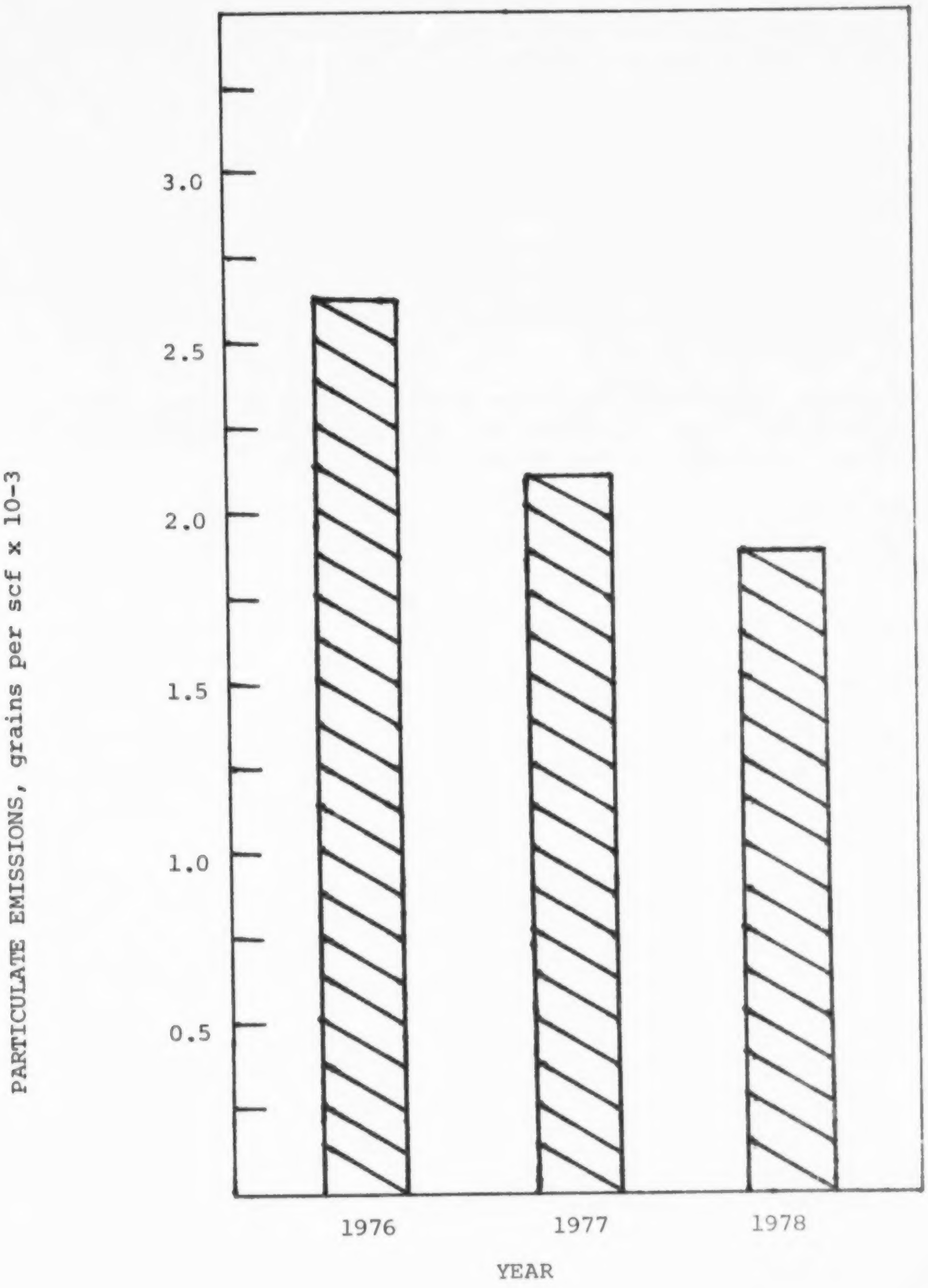

FIGURE 9. Secondary control system particulate emissions.

over the past three years. Such an achievement was accomplished by increasing the hood collection efficiency of generated particulate effluent. This involved increased supervision, maintenance and operator education efforts to assure rapid repairs, frequent equipment adjustments and operator care and attention to prevent hood damge during pot operations using automated pot working machines and manned pot tending machines.

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EMISSION CONTROL IN VERTICAL SODERBERG FURNACES

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ABSTRACT

A practical vertical stud Soderberg anode for aluminum reduction cells was developed in the late 1930's. The rigid anode casing allows the fixture of a collection skirt that captures the cell off-gases in a small volume very near the working face of the anode. Potential capture rate is somewhat less than that inherent in hooded prebake and side pin Soderberg cells. Sources of particulate and size distribution are not significantly different from those of the other types of reduction cells. Overall particulate control may require primary (pot gas) and secondary (room air) control devices. Several control schemes are discussed.

(We regret that the manuscript for this paper was not received for publication in this proceedings for reasons beyond our control.)

EXPERIENCE WITH THIRTY-ONE WET ELECTROSTATIC PRECIPITATORS ON HSS ALUMINUM POTLINES

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ABSTRACT

Thirty-one MikroPul wet electrostatic precipitators are presently collecting emissions from the six horizontal stud Soderberg potlines at the Reynolds Metals Company's Reduction Plant in Longview, Washington. Gaseous fluoride and particulate collection efficiencies of the precipitators are discussed. Also described is a major operational problem with the precipitators and its effect on collection efficiencies.

INTRODUCTION

The first precipitator, hereinafter called the pilot model, began operation in October, 1971. It fulfilled its purpose as a pilot model by being tested, modified and many times remodified. The information gained from it was used in the design of the subsequent 30 precipitators. The first of these began operation in May, 1973, with the last starting up in July, 1975.

All precipitators are preceded by cyclonic spray towers which scrub and cool the pots' emissions from about 240° F. to about 110° F. The flow rate of the alkaline liquor through the spray towers is about 180 gpm.

Six of the 31 precipitators have a design gas flow of 50,000 cfm at 70° F. and the remaining 25 are designed for 100,000 cfm at 70° F.

Inlet and outlet particle and fluoride concentrations listed in this paper were measured by methods which are modified versions of EPA Methods 5 and 13, respectively. These methods have been used concurrently with the State of Washington's EPA approved compliance test methods and test results were comparable. For comparative purposes and additional data the reader is referred to the particle size and concentration measurements conducted by Southern Research Institute on a Longview Plant precipitator.[1]

Figure 1 shows a cross-sectional view of a typical precipitator. The channel-shaped baffles, parallel collecting plates and rigid-type discharge electrodes are sprayed continuously with an alkaline liquor.

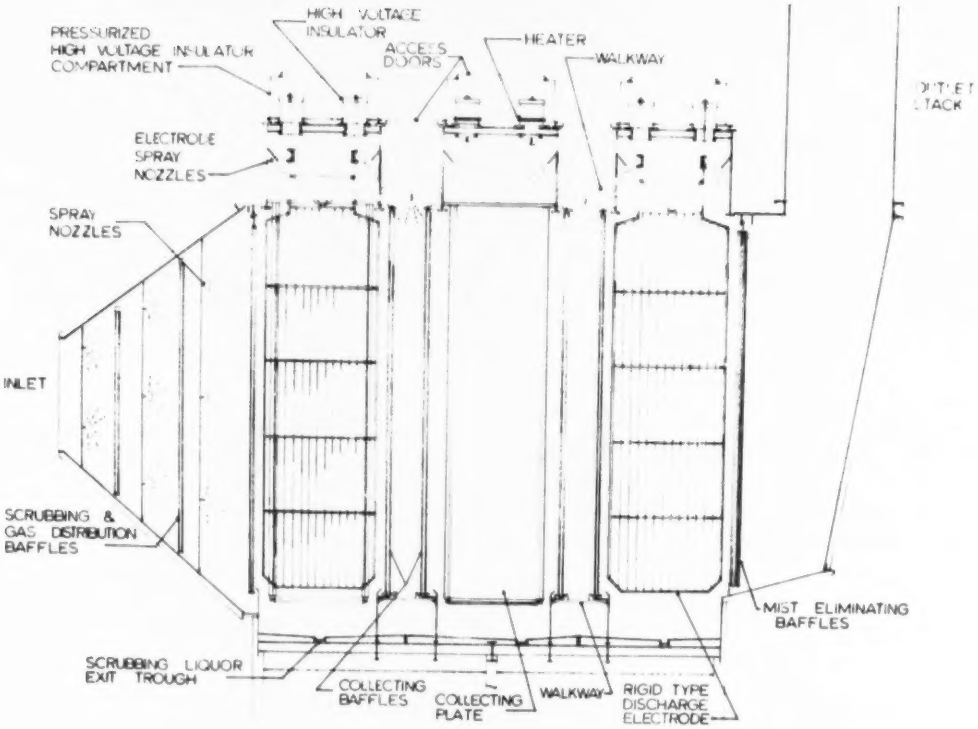
PRESSURIZED HIGH VOLTAGE INSULATOR COMPARTMENT
HIGH VOLTAGE INSULATOR
ACCESS DOORS
HEATER
WALKWAY
OUTLET STACK
ELECTRODE SPRAY NOZZLES
SPRAY NOZZLES
INLET
SCRUBBING & GAS DISTRIBUTION BAFFLES
MIST ELIMINATING BAFFLES
SCRUBBING LIQUOR EXIT TROUGH
COLLECTING BAFFLES
COLLECTING PLATE
WALKWAY
RIGID TYPE DISCHARGE ELECTRODE

Figure 1. Cross-sectional view of typical precipitator

The scrubbing liquor circulates through a treatment system which thickens the solids to about 30 percent for pumping to the Cryolite Recovery Plant. The clarified liquor is recirculated back to the precipitators.

Table 1 summarizes the specifications for the precipitators at the Longview Plant. For a more complete description of MikroPul's wet electrostatic precipitators and the theory of their operation, the reader is referred to the paper published by E. Bakke.[2]

Table 1. SUMMARY OF PRECIPITATORS' SPECIFICATIONS

Gas Flow	100,000 cfm @ 70° F.	50,000 cfm @ 70° F.
Gas Velocity	2.38 ft./sec.	2.38 ft./sec.
Treatment Time	10.1 sec.	10.1 sec.
Maximum Pressure Drop	1 in. w.g.	1 in. w.g.
Minimum Liquor Flow @ 60 psi	500 gpm	250 gpm
Number of Gas Passageways	28	14
Passageway Spacing	12 in.	12 in.
Number of Fields	3	3
Collecting Plate Length	6 ft.	6 ft.
Collecting Plate Height	25 ft.	25 ft.
Total Collecting Area	29,400 $ft.^2$	15,150 $ft.^2$
Number of Transformer Rectifiers	3	3
Rectifier Type	Silicon	Silicon
Wave Form	Full	Full
Minimum Output/T-R Set	60 kv, 1000 ma	60 kv, 500 ma
Primary Voltage	480 v, 60 hz.	480 v, 60 hz.
Housing Material	3/16 in. Mild Steel Plate	
Collecting Plate Material	10 ga. Mild Steel Sheet	
Discharge Electrode Material	1 in. x 1/8 in. Mild Steel Flatbar	
Baffle Material	2 in. x 1/2 in. x 14 ga. Mild Steel Channel	

The intent of this paper is to describe the collection efficiencies of the precipitators under various conditions of operation. This includes a discussion of a major operational problem with the precipitators and its effect on collection efficiencies. For a more complete description of the major problems that have occurred in operating and maintaining the Longview Plant's precipitators, the reader is referred to a paper previously published by this author.[3]

EFFECT OF GAS FLOW ON COLLECTION EFFICIENCY

Soon after startup of the pilot model precipitator a series of tests were conducted to determine its collection efficiency under conditions of increased gas flow. Gas flow was increased by diverting the emissions from additional pots through the pilot model. The results are listed in Table 2.

Table 2. INCREASED GAS FLOW THROUGH PILOT MODEL

Number of Tests Run	Gas Flow, cfm at 70° F. Average	Inlet Particulate, gr/Dscf Average	Inlet Particulate, gr/Dscf Std. Dev.	Outlet Particulate, gr/Dscf Average	Outlet Particulate, gr/Dscf Std. Dev.
3	47,500	0.0793	0.0052	0.00273	0.00018
4	52,500	0.0912	0.0104	0.00297	0.00033
5	54,600	0.0982	0.0079	0.00367	0.00052
4	61,200	0.0875	0.0110	0.00460	0.00054
4	62,300	0.1090	0.0246	0.00463	0.00037

Reynolds' specifications stated that the total particulate discharging from the pilot model's stack could not exceed 0.003 gr/Dscf with a gas flow of 48,500 cfm at 70° F. It was believed that a particulate loading greater than 0.003 gr/Dscf would exceed the 20 percent opacity standard imposed by the State of Washington on point source emissions within an aluminum reduction plant.

Table 2 clearly shows the pilot model met this specification. Gas flow was increased nine percent, to about 53,000 cfm before the 0.003 gr/Dscf was exceeded. The opacity of the pilot model's emissions did increase as gas flow increased, but even at the highest average gas flow, 62,300 cfm, opacity was no more than about 25 percent for an average outlet total particulate loading of 0.00463 gr/Dscf.

The data from these tests were factored into the design of the second generation precipitators of which 21 were designed for a gas flow of 100,000 cfm at 70° F. Results from increased gas flow tests on one of these precipitators are listed in Table 3 and plotted in Figure 2.

Table 3. INCREASED GAS FLOW THROUGH 100,000 CFM PRECIPITATOR

Number of Tests Run	Gas Flow, cfm at 70° F. Average	Inlet Particulate, gr/Dscf Average	Inlet Particulate, gr/Dscf Std. Dev.	Outlet Particulate, gr/Dscf Average	Outlet Particulate, gr/Dscf Std. Dev.
3	104,600	0.09534	0.01689	0.00155	0.00004
3	126,000	0.08924	0.02124	0.00314	0.00036
3	139,800	0.06814	0.01330	0.00471	0.00046

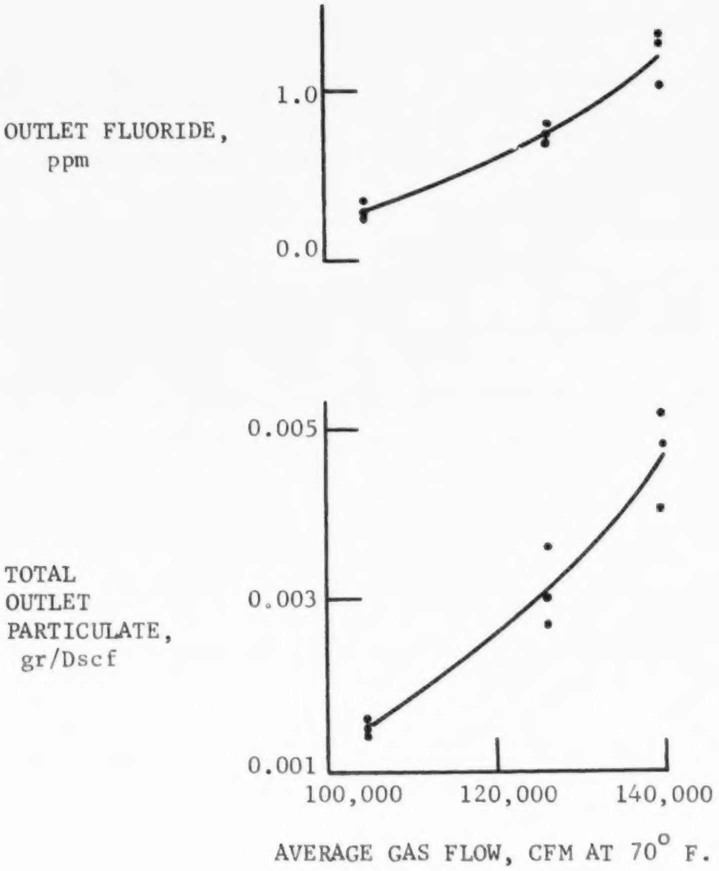

Figure 2. Increased gas flow through 100,000 cfm precipitator

Figure 2 shows that the 100,000 cfm precipitator has a greater capacity for collecting particulate from increased gas flows than does the pilot model. Its gas flow can be increased about 25 percent, versus the pilot model's nine percent, before the 0.003 gr/Dscf is exceeded. Opacity at the highest average gas flow listed in Table 3, 139,800 cfm, was about 30 percent for an average outlet total particulate loading of 0.00471 gr/Dscf.

As a matter of interest, outlet fluorides are also plotted in Figure 2.

Inlet emissions to the pilot model and 100,000 cfm precipitator, during the increased gas flow tests, were not prescrubbed. The collector electrode area inside both precipitators is about 295 $ft.^2$/1000 cfm.

There are primarily two reasons why the 100,000 cfm precipitator has a greater collecting capability than the pilot model. Its electrode alignment and gas distribution are far better.

Electrode alignment and gas distribution inside the pilot model were found to be very inadequate. Modifications were made soon after startup to improve both, but results were generally unsatisfactory. These problems were specifically addressed in the design and erection of succeeding precipitators and both electrode alignment and gas distribution were significantly improved.

EFFECT OF FIELD REDUCTION ON COLLECTION EFFICIENCY

A series of tests were conducted on the first 100,000 cfm precipitator to determine if three fields were required to meet the specified 0.003 gr/Dscf. The results are listed in Table 4.

Table 4. REDUCED FIELDS INSIDE 100,000 CFM PRECIPITATOR

Number of Tests Run	Number of Fields Operating	Outlet Particulate gr/Dscf Average	Outlet Particulate gr/Dscf Std. Dev.	Outlet Fluoride, ppm Average	Outlet Fluoride, ppm Std. Dev.
3	3	0.00117	0.00009	0.18	0.03
4	2	0.00205	0.00030	0.55	0.03
4	1	0.00682	0.00049	1.22	0.15

Test results show that with one field de-energized the average total outlet particulate was only 0.00205 gr/Dscf which, of course, easily complies with the 0.003 gr/Dscf specification. Opacity of the emission was about ten percent. However, the specification was significantly exceeded with two fields de-energized when the average total outlet particulate was 0.00682 gr/Dscf and opacity was about 35 percent.

After reviewing the results from these tests, it was tempting to fabricate future precipitators with only two fields. However, it was decided that this would be imprudent because of probable deterioration in the precipitators' collection efficiency. This has occurred and is discussed in a later section of this paper.

Also, electrical supply or other problems periodically occur which require the shutdown of a field. This does not cause an environmental problem because the precipitator is still able to comply with air regulations. As it turns out, then, the third field is a good insurance policy.

All operating conditions such as gas flow, inlet particulate loading, etc., were similar for these tests.

EFFECT OF SCRUBBING LIQUOR FLOW RATE ON COLLECTION EFFICIENCY

The scrubbing liquor flow rate into the fields of the Longview Plant's precipitators is adjusted to a minimum--just enough to satisfactorily keep the electrodes clean. The reason for this is that as liquor pressure increases, the number of droplets spraying from the nozzles rapidly increases and their mean diameter decreases. This increases the space charge density which, in turn, decreases the corona current for a given voltage. This decrease in corona power decreases collection efficiency.

This phenomenon is clearly shown in Table 5 where the liquor flow rate was varied to the inlet scrubbing chamber and to the electrostatic fields of a non-prescrubbed precipitator. The flow rate was varied by simply adjusting the liquor pressure.

By decreasing the liquor pressure over the fields from 80 psi (test series A) to 50 psi (test series B), the average corona power increased from 7.4 to 10.2 kva. As a result, total outlet particulate decreased from 0.00458 to 0.00323 gr/Dscf.

Contrary to earlier predictions, this reduction in liquor flow did not adversely affect fluoride collection. Outlet fluorides decreased from 0.97 to 0.71 ppm. This indicates that any loss of gaseous fluoride collection efficiency because of reduced liquor flow over the fields is more than offset by the increased collection efficiency of particulate fluorides.

Test series C goes a step further in liquor reduction. Liquor pressure to the inlet scrubbing chamber was also reduced from 100 psi to 50 psi. As a result, total outlet particulate increased from 0.00323 to 0.00353 gr/Dscf, but the major change was in the outlet fluorides which increased from 0.71 to 0.99 ppm. The inlet scrubbing chamber is the major area for contact between the alkaline liquor and gaseous fluorides and any reduction in its liquor flow will significantly affect fluoride collection.

The gas flows listed in Table 5 are high, about 34-38 percent greater than the 100,000 cfm for which the precipitator was designed. This precipitator and nine others were initially installed without prescrubbers. The fans for these precipitators were intentionally oversized so that if prescrubbers were installed, there would still be a minimum gas flow of 100,000 cfm.

The ratios of condensable hydrocarbons to solid particulate in both the inlet and outlet grain loadings listed in Table 5 are typical. The magnitude of the ratios will vary from test to test, but the pattern of greater solid particulate in the inlet, and greater condensable hydrocarbons in the outlet is typical.

This is well illustrated in Figures 3 and 4. The data in these figures is from tests on non-prescrubbed precipitators.

As would be expected, the inlet condensable hydrocarbon fraction is relatively constant as shown in Figure 3. There are no changes in the anode baking process that would cause a significant change in hydrocarbon emission rate.

The same cannot be said for solid particulate which are by far the

TABLE 5. EFFECT OF LIQUOR FLOW RATE ON COLLECTION EFFICIENCY

Test Series	A		B		C	
Inlet Scrubbing Chamber's Liquor Pressure, psi	90		100		50	
Electrostatic Fields' Liquor Pressure, psi	80		50		50	
Total Calculated Liquor Flow Rate, gpm	835		775		640	
Average Gas Flow, cfm at 70° F.	134,800		138,400		136,100	
Number of Tests Run	3		3		3	
Average Total Corona Power, kva	7.4		10.2		8.6	
Inlet Particulate, gr/Dscf						
	Average	Std. Dev.	Average	Std. Dev.	Average	Std. Dev.
Condensable Hydrocarbons	0.01309	0.00029	0.01279	0.00321	0.01198	0.00072
Solid Particulate	0.04218	0.00198	0.03793	0.00935	0.04104	0.00608
Total Particulate	0.05527	0.00178	0.05072	0.01232	0.05302	0.00549
Inlet Fluorides, ppm	37.60	1.71	49.57	18.46	39.53	2.47
Outlet Particulate, gr/Dscf						
Condensable Hydrocarbons	0.00261	0.00010	0.00177	0.00027	0.00184	0.00006
Solid Particulate	0.00197	0.00016	0.00146	0.00017	0.00169	0.00006
Total Particulate	0.00458	0.00021	0.00323	0.00044	0.00353	0.00011
Outlet Fluorides, ppm	0.97	0.03	0.71	0.09	0.99	0.01
Average Efficiency, %						
Condensable Hydrocarbons	80.1		86.2		84.6	
Solid Particulate	95.3		96.2		95.9	
Total Particulate	91.7		93.6		93.3	
Fluorides	97.4		98.6		97.5	

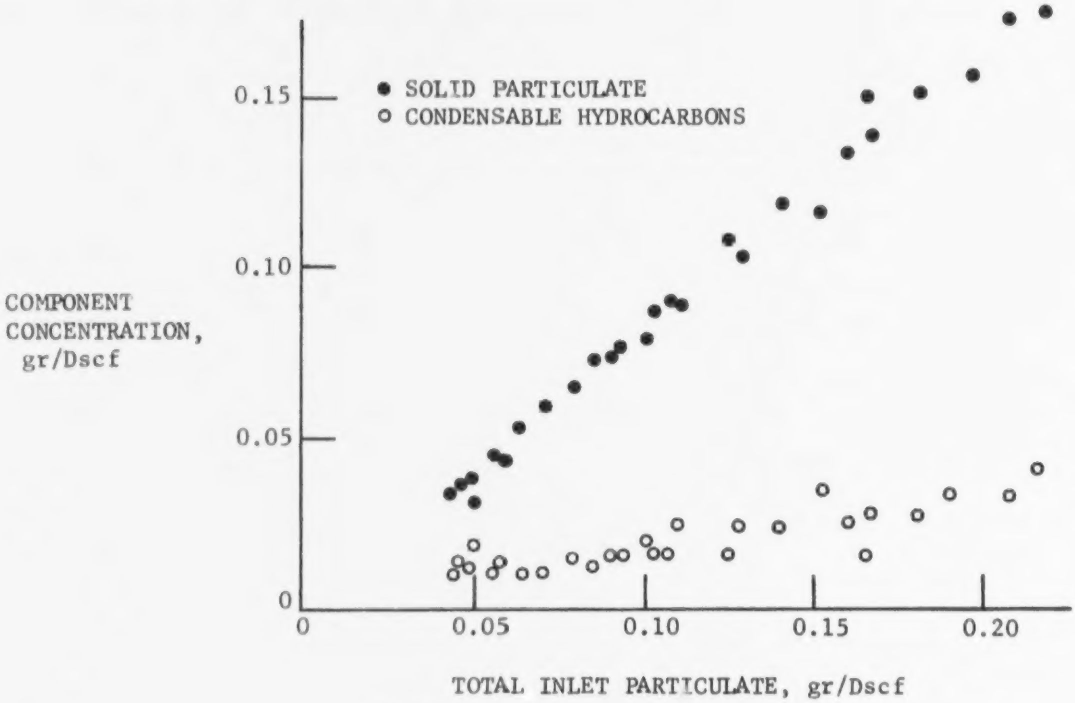

Figure 3. Composition of inlet particulate

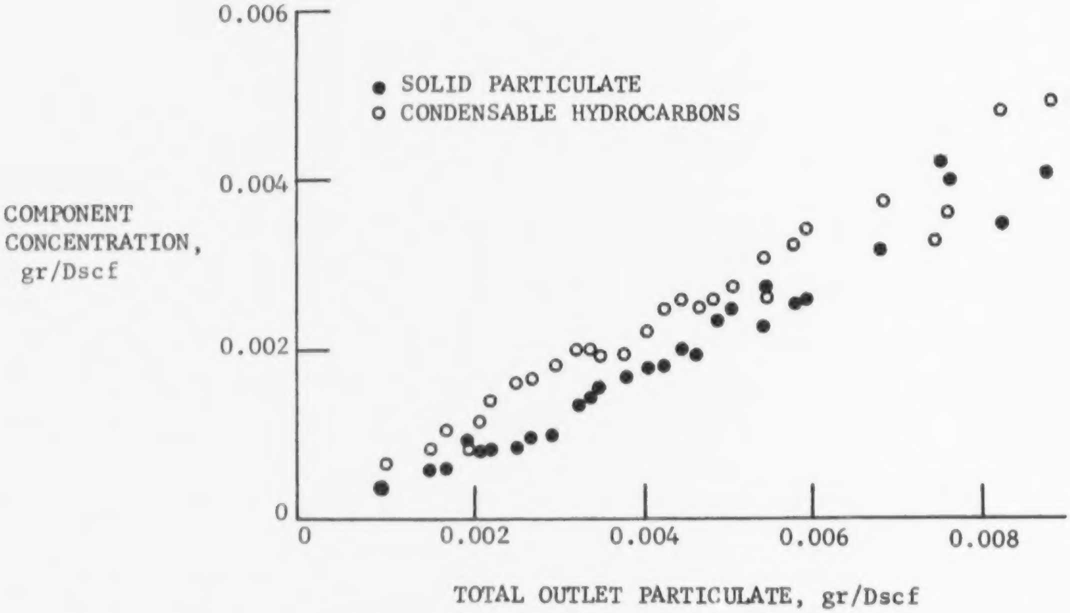

Figure 4. Composition of outlet particulate

major contributor to an increase in total inlet particulate. Routine operations on the pots such as "crustbreaking", anode "flex raises" and "pin and channel pulls" will have a very significant effect on the solid particulate emission rate.

The reverse is true for outlet emissions from the precipitator. With few exceptions, Figure 4 shows that the condensable hydrocarbon fraction is the major contributor to the total outlet particulate. This is to be expected because the hydrocarbon droplet has a lower dielectric constant than the solid particle. As a result the saturation charge on the hydrocarbon droplet will be less, making the droplet more difficult to collect.

The uniformity of the ratio of condensable hydrocarbons to solids with increasing total outlet particulate is interesting to note in Figure 4.

EFFECT OF SCALE ON COLLECTION EFFICIENCY

There were concerns about emissions from the pots accumulating on the electrodes because of the reduction in liquor over the fields. However, this problem never occurred.

What has occurred is a buildup of scale on the electrodes and inlet gas distribution and scrubbing baffles. This scale is a densely packed, tightly adhering and gray-brown layer that covers all surfaces which are wetted by the scrubbing liquor.

It is not to be confused with the buildup of constituents in the emissions from the pots. This buildup is a blackish-colored deposit, high in hydrocarbons, that can be washed off by the spray discharging from the precipitators' spray nozzles. In fact, it occurs because of plugged nozzles or nozzles not discharging enough liquor in an area.

The formation of scale was no surprise because the recirculating scrubbing liquor that passes through each precipitator is alkaline (pH 8.8 - 9.4), with a large content of dissolved solids (40,000 - 50,000 ppm). It was the scale's rapid rate of formation that was not anticipated.

The magnitude of the scale problem can be divided into separate categories--prescrubbed and non-prescrubbed precipitators. Until recently, ten 100,000 cfm precipitators were not preceded by prescrubbers as the other 21 precipitators were. The rate of scale buildup inside these non-prescrubbed precipitators was far greater than the rate inside the prescrubbed precipitators. This is well illustrated in Table 6.

The reason for the greater buildup of scale inside the non-prescrubbed precipitators versus the prescrubbed precipitators and the explanation for the scale thickness gradient from inlet to outlet, that is seen in Table 6, are discussed in a previous paper by this author.[3] The paper also describes the scale's composition and the chemical reactions involved in its formation.

Scale thicknesses of the magnitude shown in Table 6 for non-prescrubbed precipitators have a very significant effect on precipitator operation. Gas flows decreased up to 22 percent because the gaps between the inlet baffles were blinded with scale. The resulting gas pressure against the baffles caused many of them to bow. Corona currents dropped more than 50 percent because of the thick scale on the discharge electrodes.

Table 6. SCALE THICKNESS AFTER TWO YEARS OF OPERATION

Component	Scale Thickness, Inches Non-Prescrubbed Precipitators	Prescrubbed Precipitators
Inlet Baffles	1 1/2	3/8
Inlet Field Electrodes	3/4	3/8
Middle Field Electrodes	1/2	1/4
Outlet Field Electrodes	1/4	1/8

As a result of these effects, cleaning the non-prescrubbed precipitators was necessary. The most successful method to remove scale has been to vibrate the discharge electrodes and baffles with pneumatic chipping hammers. The scale is brittle when it dries and will readily shake loose. Unfortunately, the collecting plates are too stiff to vibrate the scale loose. Even so, Table 7 clearly shows the dramatic change produced by cleaning the scale from the inlet gas distribution and scrubbing baffles and the discharge electrodes inside a precipitator. Corona power increased by 77 percent.

As a result of this increase and improved gas distribution, total outlet particulate decreased 63 percent and fluorides decreased 54 percent--even though there was a 12 percent increase in gas flow.

DECREASE OF PRECIPITATORS' COLLECTION EFFICIENCY SINCE STARTUP

As would be expected, the collection efficiency of the Longview Plant's precipitators has decreased since their startup. This is graphically illustrated in Figure 5 where outlet fluorides and total outlet particulate are plotted versus time since startup. The data is from tests conducted on the first prescrubbed 100,000 cfm precipitator to begin operation May 9, 1973.

From this date to November, 1978, five and one-half years later, Figure 5 shows the total outlet particulate increased from about 0.00080 gr/Dscf to 0.00189 gr/Dscf. However, the current average, 0.00189 gr/Dscf, is still well below the 0.003 gr/Dscf that was originally specified by Reynolds for satisfactory performance. Also, the present plume opacity is less than ten percent.

The major reason for the increased total outlet particulate is the increased scale buildup which has decreased corona power by about 73 percent since startup--from about 72.2 kva to about 19.7 kva.

Table 7. EFFECT OF SCALE ON COLLECTION EFFICIENCY

	Before Cleaning			After Cleaning	
Average Gas Flow, cfm at 70° F.	141,300			158,000	
Average Total Corona Power, kva	6.6			11.7	
Inlet Particulate, gr/Dscf	Average	Std. Dev.	% Decrease	Average	Std. Dev.
Condensable Hydrocarbons	0.02517	0.00019		0.02544	0.00105
Solid Particulate	0.09591	0.00694		0.12977	0.03416
Total Particulate	0.12108	0.00713		0.15521	0.03521
Inlet Fluorides, ppm	48.61	1.53		59.80	14.25
Outlet Particulate, gr/Dscf					
Condensable Hydrocarbons	0.00549	0.00030	66.5 →	0.00184	0.00002
Solid Particulate	0.00512	0.00000	59.2 →	0.00209	0.00000
Total Particulate	0.01061	0.00030	63.0 →	0.00393	0.00002
Outlet Fluorides, ppm	3.26	0.05	54.3 →	1.49	0.03
Average Efficiency, %					
Condensable Hydrocarbons	78.2			92.8	
Solid Particulate	94.7			98.4	
Total Particulate	91.2			97.5	
Fluorides	93.3			97.5	

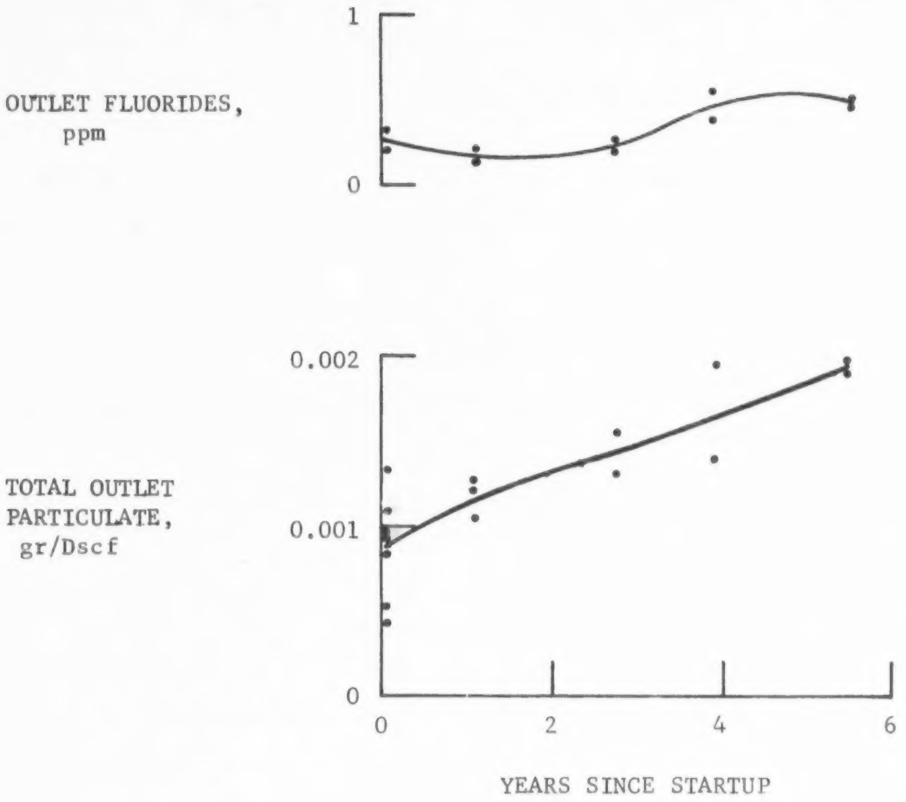

Figure 5. Increase in outlet emissions since startup

PRECIPITATORS' ELECTRICAL CHARACTERISTICS

Typical V-I curves for the three fields inside a 100,000 cfm precipitator are plotted in Figure 6. The data for these curves were obtained by manually increasing the applied voltage and recording corona current readings at the selected voltage levels. For a given voltage such as 40 kv, the outlet field has the highest current, 300 ma, followed by the middle field, 250 ma, with the inlet field having the lowest current, 200 ma.

Such a uniform difference between V-I curves is certainly not typical with all of the precipitators. The V-I curves vary considerably. However, the general pattern of increasing corona current toward the outlet is typical and is explained by the higher space charge density that results from the higher particulate loadings in the inlet field.

This same effect on current because of space charge density shows up

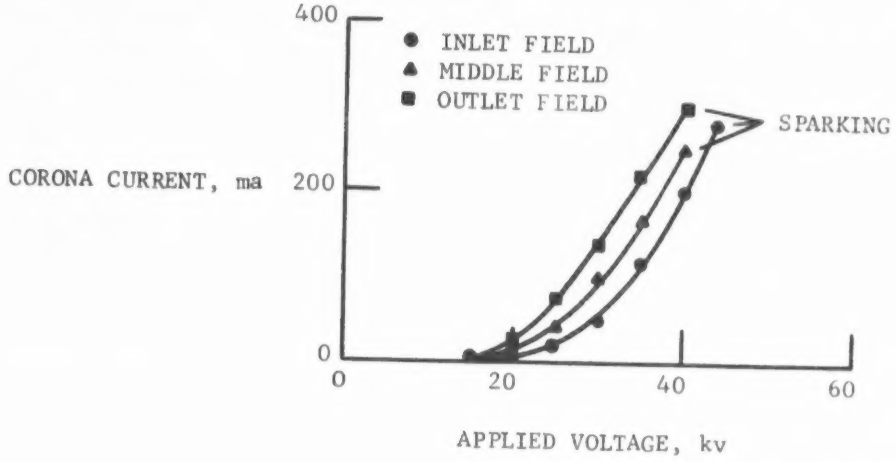

Figure 6. V-I curves for three fields inside a 100,000 cfm precipitator

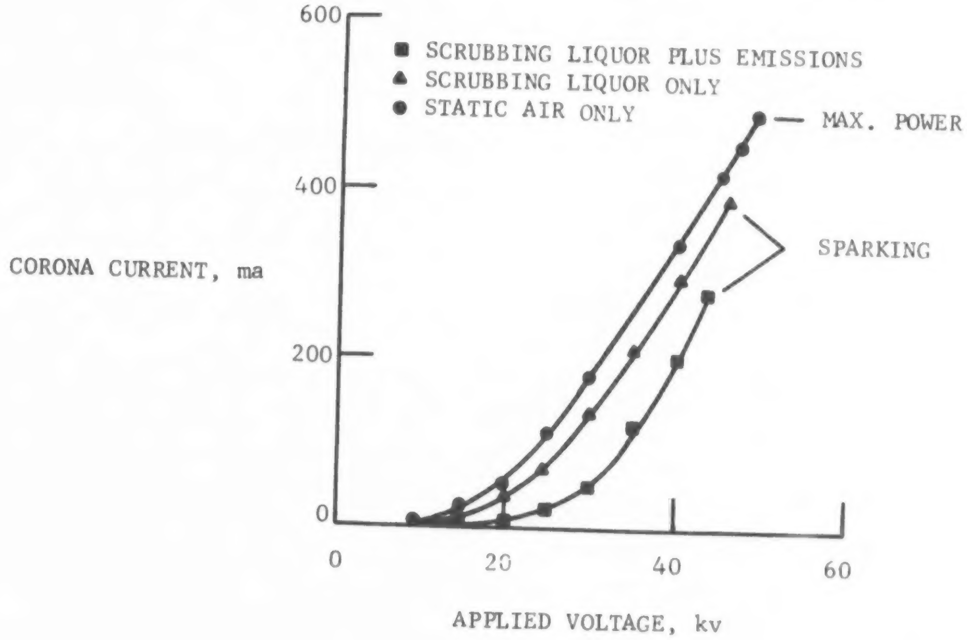

Figure 7. V-I curves for an inlet field during varying conditions of particulate loading inside a 100,000 cfm precipitator

in Figure 7 which plots V-I curves for a precipitator while energized under three different conditions: (1) static ambient air only, (2) scrubbing liquor only, and (3) scrubbing liquor plus potroom emissions. For a given voltage, such as 40 kv, the highest current, 355 ma, occurs with the static ambient air condition where, of course, the concentration of particulate is much less than in the other two conditions.

The average and range of voltage and current readings from meters on the field control panels are listed in Table 8. These readings are from 100,000 cfm and 50,000 cfm prescrubbed precipitators soon after their startup.

Table 8. PRIMARY AND SECONDARY ELECTRICAL READINGS FOR 50,000 AND 100,000 CFM PRECIPITATORS SOON AFTER STARTUP

Field	Meter		Readings For 50,000 cfm Precipitators	Readings For 100,000 cfm Precipitators
Inlet	Prim. volts	- Average	276	295
		- Range	230-305	270-320
	Prim. amps	- Average	43	88
		- Range	32-60	65-100
	Sec. kv	- Average	49	51
		- Range	46-53	46-56
	Sec. ma	- Average	149	281
		- Range	120-200	220-420
Middle	Prim. volts	- Average	287	312
		- Range	200-360	280-390
	Prim. amps	- Average	61	106
		- Range	34-90	100-110
	Sec. kv	- Average	49	47
		- Range	44-54	42-53
	Sec. ma	- Average	236	376
		- Range	140-310	320-520
Outlet	Prim. volts	- Average	269	323
		- Range	230-300	290-350
	Prim. amps	- Average	61	108
		- Range	48-71	95-130
	Sec. kv	- Average	44	49
		- Range	42-46	45-54
	Sec. ma	- Average	243	420
		- Range	175-320	350-580

The magnitude of the readings is less now because of the previously discussed scale buildup.

The total average power requirements of a 50,000 and 100,000 cfm precipitator are listed in Table 9.

Table 9. PRECIPITATORS' POWER REQUIREMENTS

Gas Flow, cfm	Item	Power Requirement, kw
100,000	Three Fields	36
	Twelve Insulator Heaters	18
	One Insulator Positive Pressure Blower, 3 hp	1.3
	Two Potroom Emission Fans, 125 hp each	168
	Total	223.3 kw
50,000	Three Fields	13
	Twelve Insulator Heaters	18
	One Insulator Positive Pressure Blower, 3 hp	1.3
	One Potroom Emission Fan, 125 hp	84
	Total	116.3 kw

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A LOW EMISSION PROCESS FOR THE MELT TREATMENT OF ALUMINUM ALLOYS

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ABSTRACT

A low emission process has been developed for degassing, fluxing and refining molten aluminum alloys. This process utilizes a mixture of an inert gas, i.e. nitrogen or argon, with 3 to 5% dichlorodifluoromethane (CCl_2F_2). CCl_2F_2 is a non-toxic compound, commercially available under the Freon 12 trademark, among others.

The gas mixture is bubbled through the melt in coated steel pipes or graphite tubes with or without a protective molten salt cover; the exact nature of the salt depends upon the composition, especially magnesium level, of the alloy. Technically, a mixture of inert gas with 3 to 5% Freon 12 is equivalent to pure chlorine in terms of rate of removal of hydrogen and non-metallic particulates. At the same time, and in contrast with chlorine, the corresponding emissions levels are well below prevailing environmental protection standards. Because of its lack of toxicity, accidental release of Freon 12 creates no occupational hazard, as opposed to chlorine. Also, since Freon 12 completely breaks down at molten aluminum temperature, none is released into the atmosphere under normal operating conditions.

Introduction

Untreated aluminum alloy melts commonly contain various impurities which can be quite detrimental to the quality of the ingots and products fabricated therefrom. Most troublesome are: dissolved hydrogen, non-metallic inclusions, principally aluminum oxide, and alkali metals such as sodium.

To control these impurities, the most common approach consists in fluxing the melt with a gas stream in order to remove the hydrogen and float up a portion of the non-metallic inclusions. A molten metal filter, either combined with or following the fluxing operation, serves to eliminate most of the remaining inclusions. The alkali metal level is controlled by using an active gas for the fluxing operation.

This melt treatment is carried out either as a batch process in the melting or holding furnace or as an in-line process in the transfer trough between the furnace and the casting station. Any melt treatment must obviously be technically effective while meeting environmental and occupational safety standards at minimum energy consumption levels.

Several in-line processes have been developed that appear satisfactory from both a melt cleanliness and emissions standpoint. Examples include Alcoa's 469 (1) and flux washing (2) processes, Linde's spinning nozzle inert flotation system (3-5) and British Aluminium's fumeless in-line degassing and cleaning method (6). These processes fit well in new cast shop facilities specifically designed to accommodate them. However, because of large space requirements, relatively high capital cost and energy consumption, they are not universally applicable. Therefore, batch melt treatment in the furnace using simple coated steel pipes or graphite tubes to introduce a fluxing gas into the melt are still in favor. Chlorine has long been considered the most efficient fluxing gas in this application. However, because of its toxicity as well as the large amount of HCl and Al_2O_3 emissions it creates, alternate gases have had to be considered, initially diluting chlorine with nitrogen, and later developing the use of N_2-CO-Cl_2 (7) gas mixtures. Although these gas mixtures reduce stack emissions, they are not as effective as Cl_2 in removing hydrogen and non-metallic particulates from aluminum and aluminum alloy melts.

These considerations, coupled with indications obtained elsewhere (8) of the degassing effectiveness of Freon 12, have led to the work reported here, which culminated in the development of an industrial, low pollution, non-toxic inert gas-Freon 12 degassing process (9-10) for aluminum alloy melts.

Description of the Inert Gas-Freon 12 Melt Treatment Process

The inert gas-Freon 12 treatment consists of fluxing aluminum alloy melts with a small amount (3 to 5%) of Freon 12 (CCl_2F_2) diluted in an inert gas such as nitrogen or argon. A thin layer of molten halide salts designed to enhance degassing efficiency is used when nitrogen is employed as the carrier gas. A NaCl-KCl salt with a small amount of fluoride salt is preferred because of cost considerations for aluminum alloys containing less than 1% Mg. A NaCl-free salt flux consisting primarily of $MgCl_2$ and KCl is preferred for alloys containing more than 1% Mg to enhance sodium removal.

In practice, the inert gas-Freon process is used for conventional batch fluxing, in either melting or holding furnaces; a simple gas mixing device is all the equipment that is required, in addition to the piping and degassing tubes conventionally available for gas fluxing.

Briefly stated, this process is technically equivalent to pure chlorine fluxing in removing hydrogen, non-metallic particulates and sodium from aluminum alloys. At the same time, it reduces stack emissions by a substantial factor over chlorine, allowing the cast shop operator to meet existing air pollution standards. Freon 12 is non-toxic so that even its accidental release in the plant will be of no consequence to the workers from an occupational safety standpoint. In the melt, Freon 12 thermally decomposes completely into carbon, fluorine and chlorine; therefore, no Freon 12 is released into the atmosphere. Chlorine and fluorine subsequently react with aluminum and other elements such as magnesium to form various compounds.

AlF_3 and/or MgF_2 are solids at molten aluminum temperature and remain on the surface of the melt. The $MgCl_2$ formed is a liquid and also remains on the surface of the melt. $AlCl_3$, which has a low sublimation temperature is in the gaseous phase, and reacts with moisture in the furnace atmosphere or stack to form hydrated Al_2O_3 and HCl. Also, a very small amount of AlF_3 and MgF_2 hydrolyze to form minute amounts of HF.

Basic Characteristics of Freon 12

The following presents some of the basic characteristics of Freon 12 as they relate to its environmental suitability as a molten aluminum treatment gas. Dichlorodifluoromethane (CCl_2F_2) is a fluorocarbon widely used as a refrigerant. It is available commercially under various trade names, one of the most common being Freon 12, a trademark of the original manufacturer, E. I. duPont de Nemours. Freon 12 has very low toxicity, rated in Group 6 by the Underwriter's Laboratories Report MH-2375,(11) with a threshold limit value of 1000 ppm (12) which is the allowable exposure to time weighted concentrations for a 7 or 8 hour work day and 40 hour work week. By comparison, the threshold limit value for chlorine is 1.0 ppm (13). Accidental release of Freon 12 in the work environment would therefore be of no consequence to the health of the workers.

is 9 feet long and is elliptical, with a major axis of 18 feet 6 inches and minor axis of 7 feet 6 inches. These dimensions provide a 3-1/3-inch clearance between the movable and fixed hoods. There are mating plates on the lower end of the fixed hood and on the top of the movable hood. The lower end of the movable hood is fitted with a 12-inch asbestos-type curtain that follows the elliptical perimeter to form a seal with the gate hood. The movable hood is constructed of 3/8-inch carbon steel with stiffeners of 7-inch channels.

- The gate hood, elliptical in cross-section, has a major axis of 16 feet 6 inches and a minor axis of 6 feet 6 inches and 9 feet long. Clearance between the fixed hood and the gate is thus 3-1/2 inches. The hood will be bolted to the gate. The plate is 3/8-inch carbon steel reinforced with 7-inch channels.

The dimensions listed above would be modified for each converter layout to provide the required clearances. Design considerations may dictate that the fixed hood is the largest unit, with a movable hood under it and the gate hood under the movable hood.

- If crane runway rail height is a problem, the smoke plenum of the secondary hooding duct system could be fitted as follows: the plenum would span the primary uptake hood and would have a secondary hood dust bin affixed on each end. The dust bins would be equipped with pneumatic dust valves and discharge pipes. No provision is made in this study for removal of dust in the dust bins. The smoke plenum for this cost study is 4 feet by 4 feet 8 inches by 21 feet. It is constructed of 3/8-inch steel with 6-inch channel stiffeners.

- The secondary hooding duct system would have an uptake from each dust bin adjacent to the converter and then pass to its main ducting header for fugitive emissions. The damper valve shown would be adjacent to the main ducting header and would be closed when the converter is out of service. Existing facilities would determine the path of retrofit. The gases go to a dust bin ahead of the fans and from there to the breeching into the main converter gas duct and to the stack. For this study, the length of the main duct runs is 600 feet.

- The fans considered in this estimate are Buffalo Forge Type 1320 BL, single-inlet, Arrangement 1, Class 3, with 145 bhp, 785 rpm, 80,000 cubic feet per minute at 200°F. There would be one fan for each converter in the plant; as many fans as are required would be tied into the system.

- Support items for the system include piping, wiring, foundations, supports for ducting every 20 feet, expansion joints, miscellaneous platforms, and walkways. Valves, fans, dust bins, and similar items are flanged for ease of maintenance.

The retrofit factor was considered as being midway between a new installation and an existing "difficult" installation.

Estimates of total installed costs are based on current (1978) costs for major components of specified sizes, as provided by equipment suppliers. Estimates of fabrication costs and installation in the Southwest are based on generally accepted engineering practices, i.e., Richardson's, Mean's, the Chemical Engineering Index, K.M. Guthrie, and data within the PEDCo organization.

A 5.0 percent contingency factor to allow for emissions testing, equipment changes, etc., is applied to the total of the direct and indirect costs. An escalation factor of 7-1/2 percent per year is used to account for increases in cost of equipment, labor, and services before and during construction. Direct capital costs include equipment, instrumentation, piping, electrical, structural, foundations, site work, insulation, and painting. Indirect capital costs include the following: engineering costs, contractor's fee and expenses, interest (accrued during construction on borrowed capital -- estimated at 9 percent per year), freight, off-site expenditures, taxes (sales tax of 4 percent of equipment cost), startup or shakedown, and spares.

Annualized costs include both operating costs and fixed capital charges and consist of: utilities; labor and fringe benefits; maintenance; plant overhead; and total fixed costs (amounting to 19.97 percent of total installed costs and consisting of depreciation over 15 years at 6.67 percent unless otherwise indicated, property insurance at 0.3 percent, property taxes at 4 percent, and interest on borrowed capital at 9 percent).

Capital costs --

Table A-1 shows the direct costs, indirect costs including one year of contingency and escalation, and total capital costs for plants containing one to nine converters without a baghouse in the fugitive emission discharge system. Table A-2 includes the costs in Table A-1 plus those for a baghouse and an appropriate increase in the fan pressure design.

Operating costs --

Operating costs include the following: operating labor at $8 per manhour, supervision at 15 percent of labor, maintenance for labor and supplies at 2 percent of total capital costs, maintenance materials at 15 percent of maintenance labor and supplies, electricity at 35 mills per kWh, plant overhead at 50 percent of operations, and payroll at 20 percent of the operating labor costs. The fixed costs include a straight-line 15 years' depreciation, 0.3 percent for insurance, 4 percent for taxes, and 9 percent for capital costs. Table A-3 lists the operating costs for a multiconverter plant without a baghouse in the fugitive emission discharge system. Table A-4 includes the costs in Table A-3 plus those for a baghouse and appropriate increases in energy and maintenance costs.

Additional handling of slag and blister ladles may cause delays in operation of the movable and swingaway converter hoods. It is estimated that a delay of 5 to 15 seconds may occur with each ladle movement, equivalent to a total delay of 3 to 10 minutes per day or a 0.23 to 0.7 percent production slowdown. This loss is calculated on an annual basis as part of the operating cost since it is negligible in comparison with other delays that are encountered, such as delay because of lack of matte, because the anode furnace cannot accept more blister copper, or because of maintenance of converters or reverberatory or electric arc furnace, etc.

Each of these systems is connected to a main discharge duct and an exhauster-type fan, which exhausts to an existing stack.

TABLE A-1. ESTIMATED CAPITAL COSTS OF SECONDARY HOODING FOR MULTICONVERTER PLANT WITHOUT BAGHOUSE

(dollars)

No. of converters	Direct costs	Indirect costs	Total costs
1	760,000	532,000	1,292,000
2	1,216,000	785,000	2,001,000
3	1,532,000	963,000	2,495,000
4	1,771,000	1,255,000	3,026,000
5	2,211,000	1,463,000	3,674,000
6	3,219,000	2,122,000	5,341,000
7	3,601,000	2,383,000	5,984,000
8	3,880,000	2,545,000	6,425,000
9	4,402,000	2,850,000	7,252,000

TABLE A-2. ESTIMATED CAPITAL COSTS OF SECONDARY HOODING FOR MULTICONVERTER PLANT WITH BAGHOUSE (dollars)

No. of Converters	Direct Costs	Indirect Costs	Total Costs
1	1,122,000	736,000	1,858,000
2	1,616,000	1,007,000	2,623,000
3	2,127,000	1,298,000	3,425,000
4	2,587,000	1,714,000	4,301,000
5	3,103,000	1,966,000	5,069,000
6	4,850,000	2,936,000	7,786,000
7	5,523,000	3,466,000	8,989,000
8	6,093,000	3,792,000	9,885,000
9	6,877,000	4,244,000	11,121,000

TABLE A-3. ESTIMATED ANNUAL OPERATING COSTS FOR SECONDARY HOODING IN A MULTICONVERTER PLANT WITHOUT BAGHOUSE
(dollars)

No. of converters	Labor and supervision	Maintenance labor, supplies, and materials	Overhead plant and payroll	Utilities	Fixed costs	Total annual costs
1	10,000	30,000	22,000	1,000	258,000	321,000
2	21,000	46,000	38,000	3,000	400,000	508,000
3	31,000	57,000	50,000	4,000	498,000	640,000
4	41,000	70,000	64,000	5,000	604,000	784,000
5	51,000	85,000	78,000	6,000	734,000	954,000
6	62,000	124,000	105,000	8,000	1,077,000	1,376,000
7	72,000	138,000	119,000	9,000	1,195,000	1,533,000
8	82,000	148,000	131,000	10,000	1,283,000	1,654,000
9	93,000	167,000	149,000	11,000	1,448,000	1,868,000

TABLE A-4. ESTIMATED ANNUAL OPERATING COSTS FOR SECONDARY HOODING IN A MULTICONVERTER PLANT WITH BAGHOUSE
(dollars)

No. of Converters	Labor and Supervision	Maintenance Labor, Supplies, and Materials	Overhead Plant and Payroll	Utilities	Fixed Costs	Total Annual Costs
1	17,000	43,000	33,000	80,000	371,000	544,000
2	28,000	60,000	50,000	122,000	524,000	784,000
3	38,000	79,000	66,000	226,000	684,000	1,093,000
4	48,000	99,000	83,000	343,000	859,000	1,432,000
5	58,000	117,000	99,000	404,000	1,012,000	1,690,000
6	72,000	179,000	140,000	523,000	1,555,000	2,469,000
7	82,000	207,000	161,000	878,000	1,795,000	3,123,000
8	92,000	227,000	178,000	1,039,000	1,974,000	3,510,000
9	103,000	256,000	252,000	1,209,000	2,221,000	4,041,000

APPENDIX B

TABLE B-1. PEDCo'S ESTIMATED HOODING EFFICIENCIES[a] (values in percent)

Hood type	Matte or hot metal addition	Blister or hot metal pouring	Skimming or rabbling	Blowing
Fixed	30-50	30-50	30-50	60-70
Fixed and movable	30-50	40-70	40-70	70-90
Fixed and swing-away	30-50	80-90	50-70[c]	80-90
Fixed, movable, and swing-away	30-50	80-90	60-80[c]	80-90
Enclosed building	90[b]-100	95[b]-100	95[b]-100	95[b]-100

[a] Most system efficiencies would be higher if air motions (i.e., open doors, man-cooling fans, monitors, etc.) could be eliminated. Skimming is removal of slag from the converter by tilting of the converter. Rabbling is removal of slag from the converter by tilting of the converter and manual use of a rake to work the molten bath.

[b] Reduced efficiency due to air motions; if doors are left open this efficiency could drop to 50%.

[c] Efficiency during skimming would be similar to blister pouring.

TABLE B-2. SUMMARY OF CURRENT FUGITIVE EMISSION CONTROL SYSTEMS

Type	Design and operation	Operational and maintenance problems	Efficiency
Monitor, Natural - U.S.	Simple design. Relies on outside air movement for removal of emissions from the area.	Haze in building during emissions; outside air movement affects time required to clear the area. Crane operator and maintenance personnel working above the converter may be required to wear face aspirators. Visible emissions in the monitor area. Maintenance in the converter area, electric overhead traveling crane and roof trusses for removal of the settled emissions other than gases.	Dependent on outside air currents and inside air motion.
Monitor, powered	Simple design. Large air movement required at the fans. Removal rate is constant.	Blind pockets or short-circuited flows could cause haze and emission buildup in the roof line area. Crane operator may require the use of a face aspirator at times. Maintenance of fans and drives. Visible emissions at each powered monitor. Maintenance in the converter area, electric overhead traveling crane and roof trusses for removal of settled emissions other than gases.	Dependent on number of monitors, fan size, building design above the converter proper; air motions.
Fixed hood with secondary emission ducting - U.S.	Clearance problems for crane hood and cables during collar pulling or matte additions. Retrofit difficulties for ducting, fans, breeching, and dust bins. Operational at all times that converters are on line. Good face and capture velocities required.	Operational damage to hood by swinging or uncontrolled electric overhead traveling crane action during matte addition or collar pulling. Maintenance is less in the converter area, electric overhead traveling crane and roof trusses due to particle buildup.	Dependent on the distance of the mouth of the fixed hood from the emission source; also on the capture and face velocity created by the fan at the mouth of the fixed hood.
Enclosed converter hood, swing-away type with fixed hood	Clearance problems for crane hood and cables during collar pulling or matte additions (fixed hood). Clearance problems for floor space relationship to the fixed hood; rugged drive mechanism needed for swing-away converter hood.	Space occupied in the aisle by the swing-away converter hood when adding matte, rabbling, or skimming could hamper crane movements. Crane must deposit the ladles for pouring or skimming and at the completion of the operation must await retraction of the hood before engaging the ladle. Maintenance of the swing-away mechanism and minimal maintenance for removal of particulate buildup that occurs during matte additions and rabbling.	Dependent on operational cycle. When pouring, blowing or slagging the emissions control woudl be good; when adding matte, or rabbling, it would be similar to the fixed hood. Air motions influence efficiency in all operations.
Enclosed building	Requires careful consideration of all openings (personnel, truck, rail, materials) to minimize air motion. Roof monitor must handle all ventilation air for workers, emissions and in-leakages. Increases building costs because of wind load design, tightness, and close-fitting openings.	All openings must be maintained constantly against excessive air infiltration. Tight siding and roofing required. Air circulation within the building for the workers and process must be carefully controlled. Intake and exhaust fans require preventive maintenance. Cleanup maintenance for settled particulates in the converter area is similar to that for a monitored system, either natural or powered.	Dependent on building tightness, air motion control, monitor exhaust capabilities.

APPENDIX C

AIR CURTAIN CONTROL TECHNIQUE FOR FUGITIVES

A method somewhat similar to secondary hooding for controlling fugitive emissions at a Peirce-Smith converter would be the use of an air curtain technique as used by Mitsubishi in Japan. This technique could also be used on other smelter operations, i.e., roaster, reverberatory furnace, etc.

This consists of steel side panels with a back and top panel, forming a partial enclosure. The front or approach side by the crane with a ladle to the converter is open. The top panel has an opening of 1.4 meters to permit the overhead crane's cables to pass into the enclosure when charging matte or removing converter slag or blister copper ladles.

Air is blown from one side of the top panel opening and collected on the other side of the opening. The air flow rate across the opening is 1000 NM^3/min. This flow plus the rising fugitives are captured and discharged to a fugitive collection system.

We developed costs using this air curtain fugitive control technique in a manner somewhat similar to that shown in Appendix A for secondary hooding systems.

Our design differs from the Mitsubishi in that the side panels were extended to contain the location area of the ladle awaiting blister copper or slag from the converter. Also partial covering was placed on the approach side (Figure 29).

The air flow across the top was set at 1000 NM^3/min but the collection side was designed with its own suction fan to handle 1,250 NM^3/min to create a flow pattern to capture the fugitives during charging, teeming, slagging, etc.

Tables C-1 and C-2 show the total installed capital and annual operating costs.

TABLE C-1. ESTIMATED CAPITAL INSTALLED COSTS FOR AIR CURTAIN TYPE HOODING WITH BAGHOUSES FOR MULTICONVERTER PLANT

No. of converters	Direct cost	Indirect costs	Total costs[a]
1	$ 372,200	$ 586,700	$1,082,000
2	736,900	926,700	1,878,000
3	987,800	1,222,200	2,495,000
4	1,365,500	1,708,800	3,470,000
5	1,621,200	2,010,200	4,099,000
6	2,111,300	2,725,500	5,460,000
7	2,688,700	3,406,300	6,880,000
8	3,242,000	4,189,800	8,389,000
9	3,591,200	4,630,300	9,280,000

[a] Includes a 5% contingency and 7-1/2% escalation for one year.

TABLE C-2. ESTIMATED ANNUAL OPERATING COSTS FOR AIR CURTAIN TYPE HOODING WITH BAGHOUSES FOR MULTICONVERTER PLANT

No. of converters	Labor & supervision	Maintenance, labor, supplies & mat'ls	Overhead plant & payroll	Utilities	Fixed costs	Total Annual costs
1	17,000	25,000	20,000	59,000	220,000	341,000
2	28,000	43,000	29,000	67,000	382,000	549,000
3	38,000	57,000	36,000	80,000	506,000	717,000
4	48,000	80,000	47,000	145,000	704,000	1,024,000
5	58,000	94,000	54,000	168,000	832,000	1,206,000
6	72,000	126,000	76,000	225,000	1,109,000	1,608,000
7	82,000	158,000	92,000	240,000	1,398,000	1,970,000
8	92,000	193,000	110,000	297,000	1,704,000	2,396,000
9	103,000	213,000	120,000	349,000	1,886,000	2,671,000

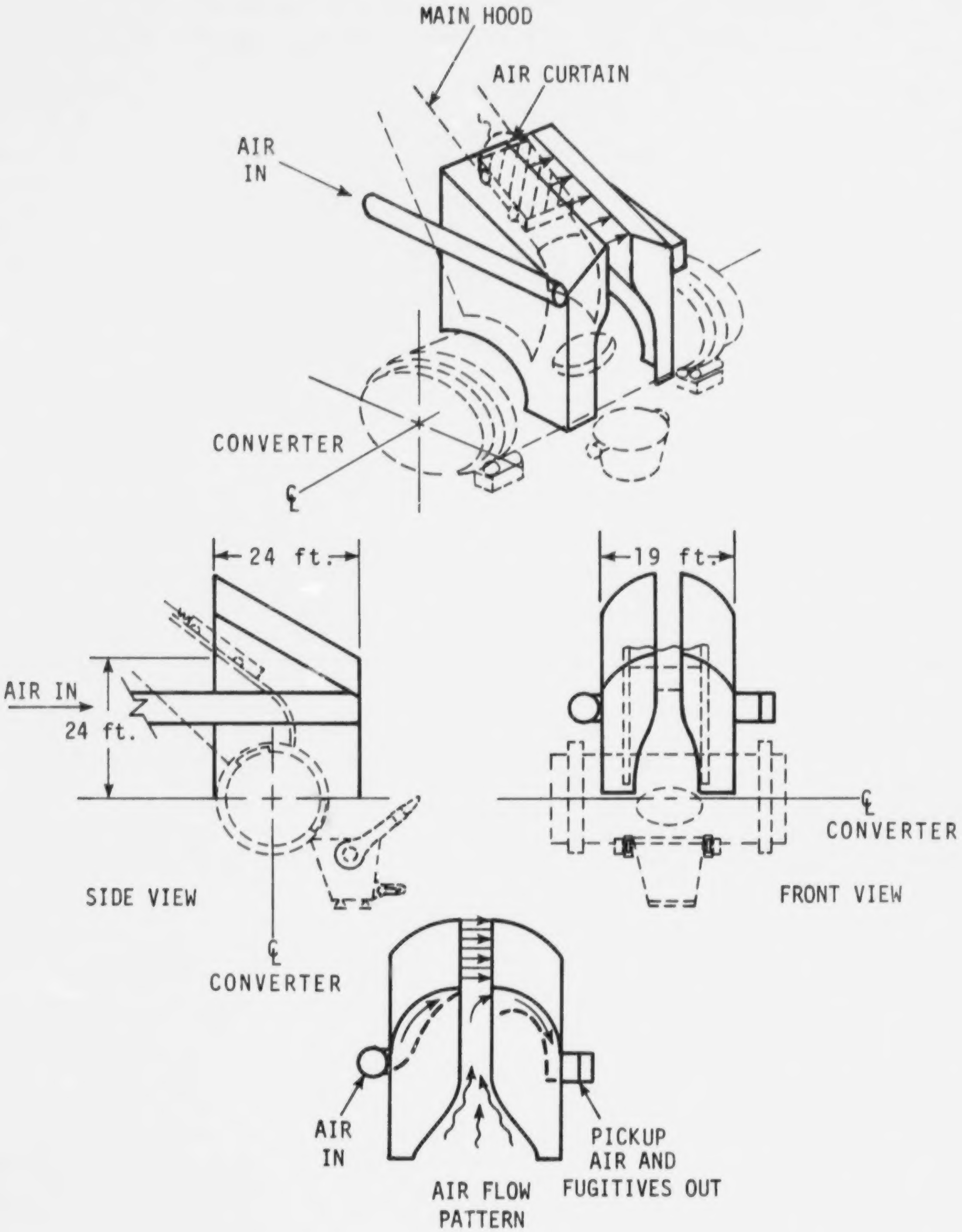
MAIN HOOD
AIR CURTAIN
AIR IN
CONVERTER
℄
24 ft.
AIR IN
24 ft.
SIDE VIEW
℄ CONVERTER
19 ft.
℄ CONVERTER
FRONT VIEW
AIR IN
AIR FLOW PATTERN
PICKUP AIR AND FUGITIVES OUT

Figure 29. Air-curtain fugitive control system.

DEMONSTRATION OF THE BERGSØE AGGLOMERATION FURNACE AND BEST MANAGEMENT PRACTICES AT A SECONDARY LEAD SMELTER

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ABSTRACT

The United States Environmental Protection Agency has recently promulgated a new National Ambient Air Quality Standard for lead at 1.5 micrograms of lead per cubic meter of air (quarterly average as measured at the fence line). This standard, coupled with the OSHA occupational safety standard of 50 micrograms lead per cubic meter of air is anticipated to have a significant impact on the primary and secondary lead industry in the United States.

This paper provides an overview of EPA's control technology study for the secondary lead industry. Preliminary findings concerning the exemplary practices utilized at a secondary lead smelter in Denmark are reviewed and the application of similar techniques to a U.S. smelter is discussed. Indications are that major contributors to the violation of this standard are fugitive sources such as piles of baghouse flue dust, raw materials storage areas, and general inattention to plant housekeeping.

INTRODUCTION

This paper describes a study being conducted by the Industrial Environmental Research Laboratory (IERL-CI) of the U.S. Environmental Protection Agency in Cincinnati. The study is a four part demonstration of fugitive emission controls and what EPA terms "best management practices" (BMP's). The objectives of the program are first, to determine what improvement can be made in lead-in-air levels both inside and ouside a smelter by retrofitting fugitive emission and instituting work practice controls (BMP's) and second, to determine the cost.

The first part of the study was conducted at the Paul Bergsøe and Son secondary lead smelter in Glostrup, Denmark. As part of an inter-agency effort, the Control Technology Assessment Branch of NIOSH and the Metals and Inorganic Chemicals Branch of EPA performed this characterization of the new secondary lead smelting technology being used at the Bergsøe smelter. The processes characterized offer potential solutions to major occupational and environmental problems associated with secondary crude lead production. Two major processes were studied: special battery (SB) smelting and flash agglomeration of the flue dust. The remainder of the program, which is also an inter-agency effort, involves: first, measuring the present lead-in-air levels at a U.S. smelter; second, retrofitting the Bergsøe dust agglomeration furnace to an existing U.S. smelter, instituting BMP's, and cleaning the smelter; and finally, measuring the reduction in lead-in-air levels both inside and outside the smelter.

This paper contains only a portion of the results obtained in the characterization study of both the Bergsøe smelter and the initial characterization of the U.S. smelter. The complete results of the Bergsøe study are available in a draft report entitled "Source Characterization of the SB Smelting Furnace." Copies can be obtained from the Cincinnati offices of EPA's Metals and Inorganic Chemicals Branch and NIOSH's Chemical Agents Control Section.

PROJECT OVERVIEW

A special report prepared in 1976 for IERL-CI identified secondary lead as an industrial segment of the secondary nonferrous metals industry posing a serious environmental threat based on the data gathered for two environmental assessment reports. As part of a larger study of the nonferrous industry, IERL-CI commissioned an evaluation of new technologies and control options for reducing process and fugitive emissions of lead. The primary goal of this undertaking was to identify new approaches to metal production which had been developed with environmental considerations given priority as key design parameters. In addition, control options readily adaptable to the industry were to be similarly sought. It was intended that, following these preliminary fact-finding studies, more detailed investigation of each important process could be undertaken.

The study of the emerging and advanced processes for secondary lead was to find solutions to the environmental and occupational problems of:

- battery breaking and consequent battery acid drainage and spillage,
- stack emissions of lead, arsenical, and antimonial particulate and fume,
- fugitive emissions such as unducted process gases and dust sources such as air pollution control residues, and raw and waste material storage piles, and
- employee exposure to lead in the workplace.

The most attractive potential solutions to the major environmental problems associated with the secondary lead industry were analyzed to identify the best available control technology. Two major processes were identified which presented opportunities for significant environmental improvement: whole battery smelting and flash agglomeration of flue dust. These processes represented the most advanced technology for controlling emissions from secondary crude lead production. Whole battery smelting offered solutions to both environmental and occupational health problems. The SB (special battery) smelting system at the Bergsøe smelter in Denmark included both environmental and occupational considerations as key design parameters. Plant layout, raw material storage and handling, process and hygiene ventilation, housekeeping, process control, and flue dust agglomeration were each included in the fundamental design. By including these design parameters, it appeared to be possible to produce crude lead without the usual environmental and occupational health problems.

Flash agglomeration of flue dust appeared to be an effective method for minimizing fugitive dust problems, improving baghouse performance, and increasing smelting capacity. The major advantages that flue dust agglomeration offered were:

- an 80 percent reduction in the specific volume of baghouse dust,
- a virtually dustless product in a form which is ideal for charging to a blast furnace,
- reduced dust loads in the blast furnace off-gas thus improving efficiencies and reducing operating costs of existing baghouses,
- increased blast furnace charge rates,
- elimination of fugitive emissions from piles of stored flue dust.

Although these claims appeared to be reasonable, they needed to be substantiated, especially at a U.S. secondary lead smelter. Actual reductions in stack and fugitive emissions which could be realized by installing an agglomeration furnace have not been quantified. There are other factors such as dust melting point and smelter capacity which could also limit the applicability of the process. To answer these questions, both source characterization and technology demonstration programs were suggested for both new and old smelters to quantify the reduction in fugitive emissions which could be achieved by using flue dust agglomeration.

The Control Technology Branch of the National Institute of Occupational Safety and Health commissioned a similar study to document and evaluate the most effective control technology available to minimize worker exposure in the secondary nonferrous smelting industry. This evaluation was accomplished by review of existing information from government and other sources as well as a series of preliminary and in-depth plant surveys.

The major result of this study was that successful control of employee exposure was almost always dependent upon an integrated control program which required the application of several different control elements, none of which worked effectively without the others. In most of the smelters surveyed, only a portion of a completely integrated control program was in use. Thus, effectiveness of controls in use in one area of the smelter was masked by contamination from other uncontrolled or poorly controlled sources. All of the exposure control elements identified were applications of basic industrial hygiene exposure control principles.

The EPA initiated an interagency agreement with NIOSH to measure fugitive atmospheric emissions and workplace employee exposure levels from secondary nonferrous smelting. These terms, synonymous expressions that the two agencies use, describe airborne pollutants that are discharged at low levels during raw material charging, slag tapping and metal pouring and are essentially uncollected by process controls. The purpose of this program was to jointly measure the levels of lead-in-air at the Bergsøe smelter and at a U.S. smelter and then evaluate the effectiveness of a combination of controls in reducing lead-in-air levels at the U.S. smelter. The controls of particular interest in this case were the agglomeration furnace, BMP's, and an improved ventilation system. The joint program resulted in considerable cost savings to the government in that a sampling program satisfying both R&D programs was fashioned at less cost than separate sampling by each agency.

RESULTS OF THE SB FURNACE CHARACTERIZATION

SB smelting is an important process because it eliminates the battery decasing step when whole batteries comprise a portion of the furnace charge. During the test period, whole batteries comprised 40 percent of the lead-bearing charge. In addition, since flue dust agglomeration is an integral part of the SB smelter, a major part of the smelter fugitive dust emissions is eliminated. It is important to note, however, that SB smelting alone is applicable only for the production of crude lead. Additional refining processes are required to produce soft lead, lead oxide, or lead alloys.

The SB smelter plant layout, raw material storage and handling facilities, process and hygiene ventilation, and flue dust agglomeration system all include both environmental emission control and occupational health considerations in their design. Also, the SB furnace is controlled at conditions producing minimum emissions. This approach has been implemented by Paul Bergsøe and Son A/S of Glostrup, Denmark.

SB Smelting Furnace

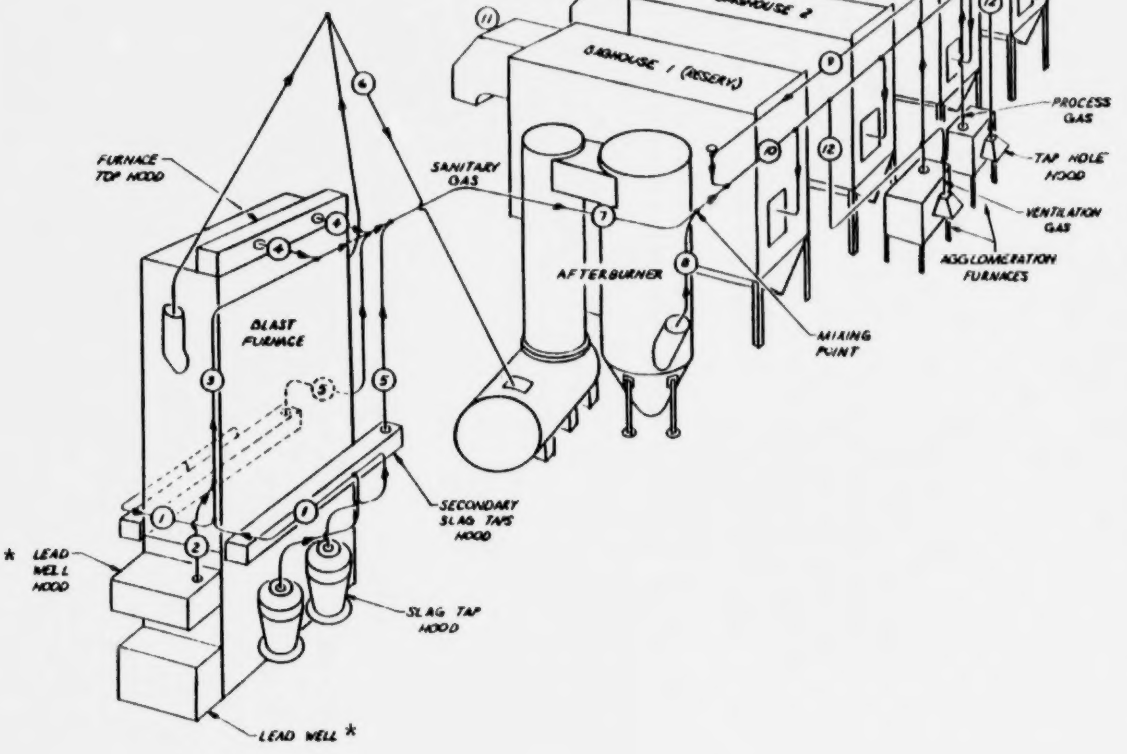
The SB smelting shaft furnace has an oblong cross section unlike most cylindrical secondary lead blast furnaces used in the United States. The construction is similar to a primary lead blast furnace. Figure 1 is a diagram of the furnace and associated gas treatment system. The furnace is constructed so as to isolate the charging floor from the bottom of the furnace. Thus, only the front-end loader operator works in a "dirty" area. However, the front-end loader does have a filtered air supply. In addition, the top of the blast furnace is provided with local exhaust ventilation to minimize fugitive emissions.

At the bottom of the furnace, local exhaust ventilation is provided for the four slag taps and the lead ladles. There are two rows of tuyeres, one on either side of the furnace, designed to use air preheated to 500°C. The tuyeres have special covers which minimize fugitive emissions during punching.

The local exhaust ventilation air and process flue gases are mixed and all gases pass to four baghouses (at 100 to 125°C). The baghouses are a Swedish design using felted polyester cloth. The dust is collected on the outside of the bag and only a mild cleaning air stream is required to dislodge the dust. This reportedly gives the bags an exceptionally long life. The baghouses were designed to operate with three running and one spare. During the test period, all four were running. The air-to-cloth ratio with four bags operating is 34.5 $m^3/hr/m^2$ (1.88 $ft^3/min/ft^2$). The effective cloth area is 4720 m^2 for four baghouses. Typical pressure drop across the baghouse is 150 mm ($\sim$6 inches) water gage.

The collected dust is conveyed in an enclosed screw conveyor system to one of two small flash agglomeration furnaces. In this patented process, the dust is melted and reduced in volume by about 80 percent. This reduction in volume increases production per square foot of furnace cross-section by reducing the dust recirculation. The agglomerated dust represents only 2 or 3 percent by weight of the furnace charge. Agglomerating the dust also reduces the dust load circulating in the gas cleaning system. The flash agglomerator furnace is oil fired and consumes approximately 7.7 liters of oil per hour.

The smelter area is paved and is wetted and swept periodically. This practice minimizes fugitive emissions normally caused by the wind blowing dry


FURNACE TOP HOOD
SANITARY GAS
BAGHOUSE 1 (RESERV)
BAGHOUSE 2
BAGHOUSE 3
BAGHOUSE 4
PROCESS GAS
TAP HOLE HOOD
VENTILATION GAS
AGGLOMERATION FURNACES
AFTERBURNER
MIXING POINT
BLAST FURNACE
SECONDARY SLAG TAPS HOOD
* LEAD WELL HOOD
SLAG TAP HOOD
LEAD WELL *
1
2
3
4
5
6
7
8
9
10
11
12

STREAM No.	STREAM NAME
1	SLAG TAP VENT
2	LEAD WELL VENT
3	①+②
4	FURNACE TOP SANITARY VENT
5	SEC. SLAG TAP VENT
6	PROCESS GAS
7	③+④+⑤
8	AFTERBURNER OFF GAS
9	AGGLOMERATION FURNACES GAS
10	⑦+⑧+⑨
11	COMBINED EXIT GAS
12	AGGLOMERATION FURNACES VENT

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Figure 1. Whole battery smelter furnace and flue gas treatment system.

lead dust in the yard. This rinse water is collected and combined with the acid drainage from cracked batteries stored in the yard and is finally treated using a soda ash precipitation process. The treated effluent is discharged to the municipal wastewater treatment facility. The sludge is withdrawn approximately once every two weeks and is charged back to the furnace for additional recovery of metals.

This smelter is serviced by two additional sewer collection systems, one for rainfall and one for sanitary sewage. The rainfall collected is used as washdown water for the smelter yard. Additional makeup cooling water is obtained from the municipal water supply. This water is softened in an ion exchange unit before use. Sanitary sewage is discharged directly to the municipal collection system.

Furnace Operation--

The operation of the SB furnace results in low stack and fugitive lead emissions rates. The relatively low blast air rate ($\sim$3500 Nm^3 air/hr) and large furnace cross-sectional area (4.0 m^2 at the tuyeres) results in a low gas velocity. This combined with the low furnace top temperatures and the absence of loose flue dust in the charge result in a low lead dust generation rate. The large furnace cross-section and small production rate allows the charge material to descend slowly through the furnace shaft. Thus the charge heats slowly and is not hot enough at the top of the furnace to generate lead fume.

This slow heating also prevents the rubber and polypropylene case material from "burning thru." Burn-thru could occur when the charge material is ignited throughout the furnace shaft rather than only in the smelting zone. This occurs more readily in the SB furnace than in conventional blast furnace because the rubber or polypropylene case material is present. It is therefore very important to control the furnace temperature both from an operating and an environmental viewpoint.

Furnace Charging--

In order to help maintain constant temperature in the furnace, the feed is carefully bedded on the chargeroom floor. A layer of coke is spread on the floor first, followed by recycle slag, batteries, plates, scrap iron and other feed materials. By doing this, each bucket of material charged to the furnace contains roughly the right amount of coke. This practice maintains a homogeneous mixture of material in the furnace and helps avoid hot spots.

Each furnace charge during the test period contained roughly the same ratio of the materials shown in Table 1. A front end loader is used to spread the feed materials on the chargeroom floor. A large floor scale is used to weigh the front end loader with a full bucket. The measurement is fed to the mini-computer which monitors the smelter operations. The weight of the empty front end loader is subtracted and the charge weight is recorded. A very accurate measurement of the charge blend is possible using this technique, typically within 1 to 2 percent of the target on major components (batteries, plates, etc.) and within 4 to 5 percent on the minor flows.

TABLE 1. TYPICAL FEED MAKEUP FOR SB SMELTING FURNACE CHARGE

Feed Material	Approximate Weight Percent	Percent of Lead Bearing Charge
Whole Batteries		
-polypropylene case	12.6	20.0
-hard rubber case	12.6	20.0
Battery Plates	31.5	50.0
Agglomerated Dust	3.2	5.0
Drosses (or battery mud	3.2	5.0
Return Slag	22.1	
Coke	5.7	
Scrap Iron	1.9	
FeO (mill scale)	6.3	
$CaCO_3$	0.9	
Total	100.0	100.0

Operating Paramters--

Scrap iron and mill scale (FeO) are added to the furnace so that a PbS-FeS matte forms between the lead bullion and slag in the furnace. Most of the sulfur input is trapped in this matte. At present, the matte is being stored on-site because of its high lead content (∿8%). It must either be sold to a primary lead smelter or disposed of in a protected landfill. Recovery of the matte lead content at a secondary lead smelter is not possible because of the sulfur dioxide emissions which would result. A primary lead smelter, however, could combine this material with the normal lead sulfide materials they process.

Oxygen enriched air is used in the SB furnace. Blast air is preheated to 500°C then mixed with oxygen prior to entering the furnace. Preheating the air reduces the amount of coke required in the furnace and allows smaller blast air rates to be used. As mentioned earlier, this helps reduce dust generation in the furnace. Blast air pressure at the tuyeres ranges from 800 to 1200 mm H_2O. The blast air rate is typically between 3200 and 3700 Nm^3/hr with between 60 and 115 Nm^3/hr of oxygen added. These rates correspond to a production rate of between 62 and 74 metric tons Pb per day.

Flash Agglomeration Furnaces

The flue dust generated by automobile battery and battery manufacture scrap smelting melts at approximately 400 to 900°C (750 to 1650°F). This is the property which makes flash agglomeration of flue dust possible. Dusts with higher melting points cannot be agglomerated using this technique without causing the low melting point materials to volatize. A special furnace was designed to take advantage of this property so that dust handling could be completely avoided.

At most secondary lead smelters, it is common practice to return flue dust directly to either the blast furnace or a reverberatory furnace. A considerable amount of this dust is also entrained in the furnace flue gas system. Agglomerating the flue dust prevents entrainment, thus reducing the load on the baghouse and improving its performance.

The Bergsøe smelter has two agglomeration furnaces serving the four SB furnace baghouses. Figure 2 is a diagram of a flash agglomeration furnace. The agglomeration furnace is fed directly from the baghouse dust hoppers via screw conveyor. The dust drops onto the furnace hearth where it melts almost instantaneously upon contact with an impinging flame. The liquid runs down the sloping hearth, through a permanently open taphole and into a cast iron vessel where it solidifies.

The transfer of flue dust is usually a dusty operation and is a source of fugitive emissions and high exposure to lead-in-air. The agglomeration furnaces completely eliminate handling of the dust, the associated occupational hazard, and fugitive emissions from flue dust storage piles provided that the agglomerated dust is stored indoors.

Tipping the contents of the cast iron vessels onto the floor is usually sufficient to break the material into lumps suitable for recharging to the blast furnace. It is simply mixed with coke and flux and loaded into the top of the blast furnace along with other charge materials.

Because less flue dust is generated, and because the volume being recycled is reduced by about 80 percent, additional material can be charged to the furnace, thus increasing the smelting rate slightly. This is one economic justification for the agglomeration furnace.

Plant Layout

A plot plan of the new Bergsøe smelter is shown in Figure 3. The old lead smelter, the lead refinery, the small cast iron department, the copper department, and the other small smelting operations are located between 100 and 400 meters southwest of the new smelter. The old smelter was built in the 1930's. Construction on the new smelter began in 1973 and the initial startup occurred in 1975.

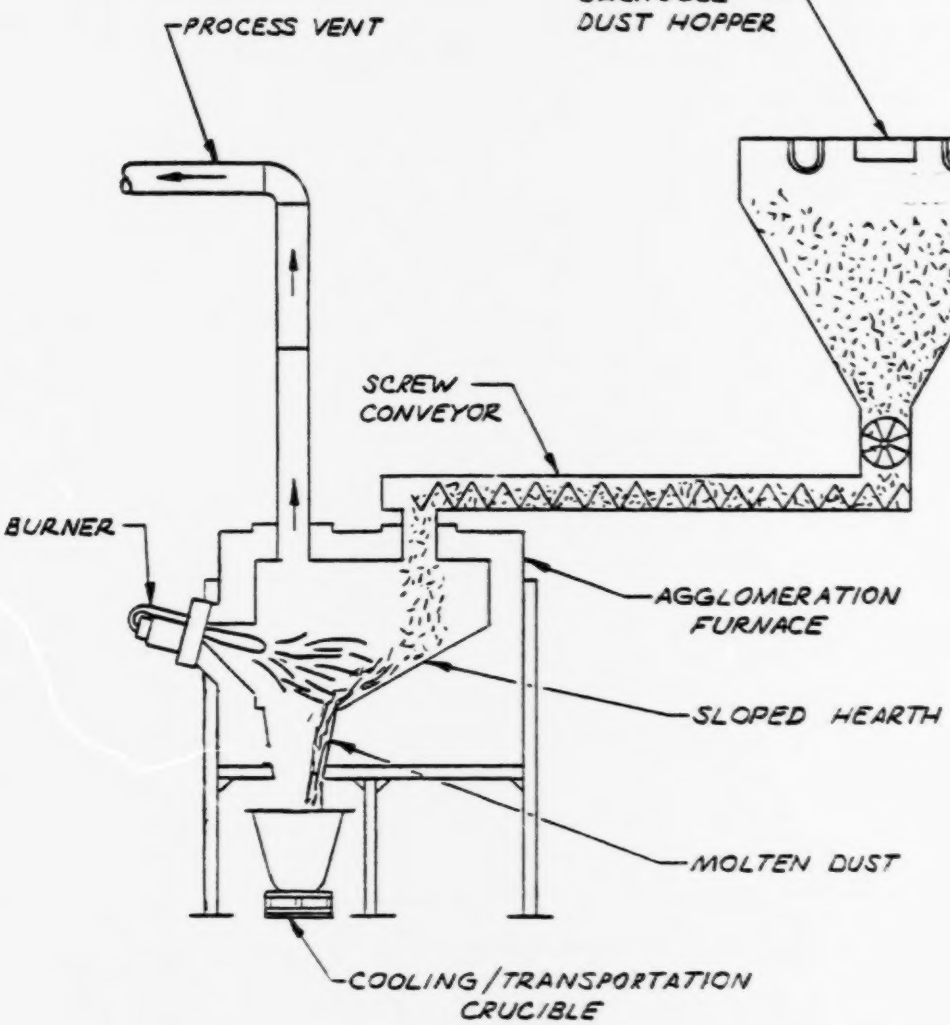
PROCESS VENT
BAGHOUSE DUST HOPPER
SCREW CONVEYOR
BURNER
AGGLOMERATION FURNACE
SLOPED HEARTH
MOLTEN DUST
COOLING/TRANSPORTATION CRUCIBLE

Figure 2. Flash agglomeration furnace.

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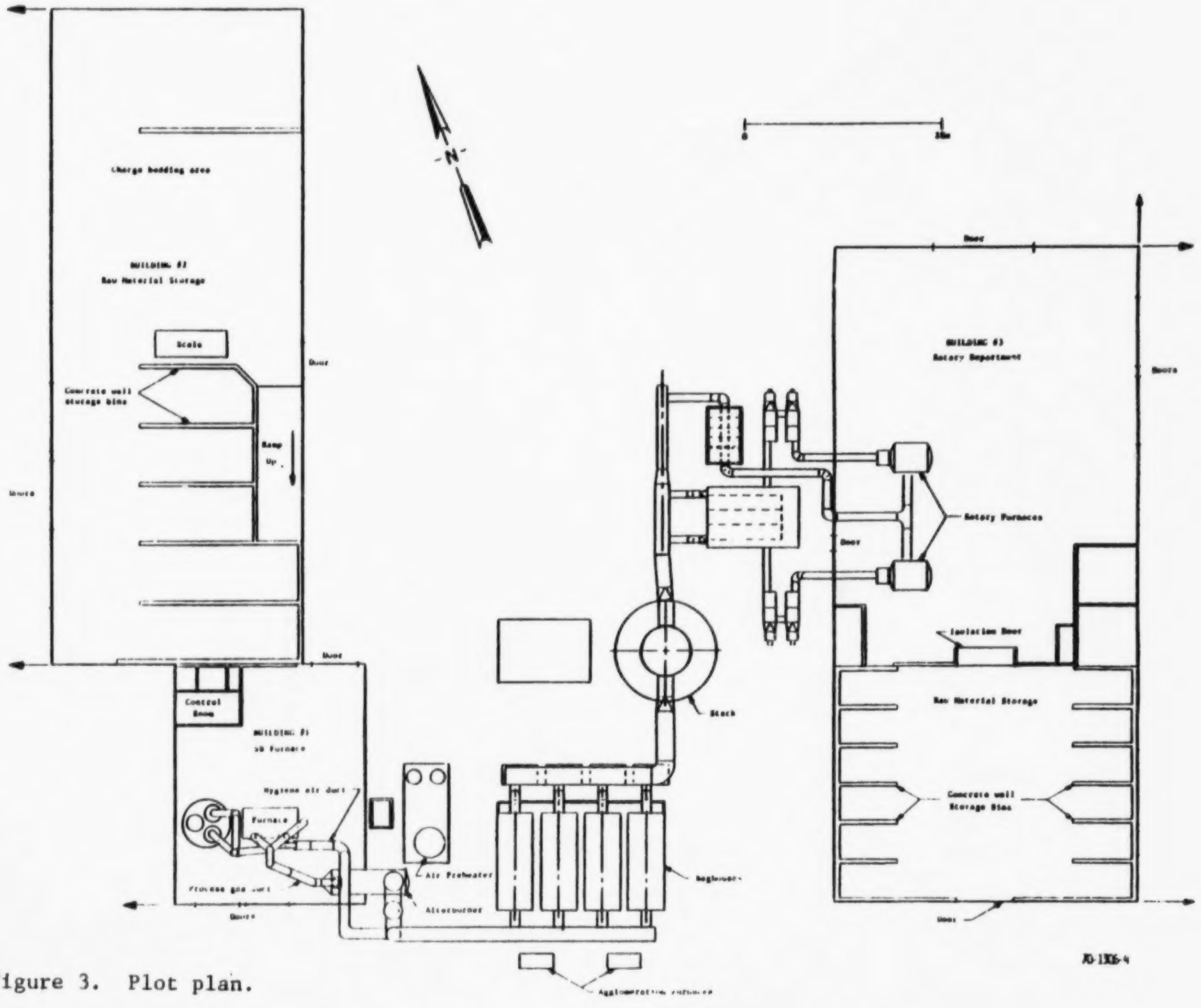
N
0
30m
Charge bedding area
BUILDING #2
Raw Material Storage
Scale
Concrete wall storage bins
Door
Ramp Up
Doors
Door
Control Room
BUILDING #1
SB Furnace
Hygiene air duct
Furnace
Process gas duct
Doors
Air Preheater
Afterburner
Baghouses
Agglomeration furnaces
Stack
Door
Rotary Furnaces
BUILDING #3
Rotary Department
Doors
Isolation Door
Raw Material Storage
Concrete wall Storage Bins
Door
JO-1306-4

Figure 3. Plot plan.

The SB furnace is located in the building labelled #1 on Figure 3. As mentioned earlier, the furnace building isolates the top of the furnace where the charging is done from the work area on the first floor. A large hood over the charge area captures emissions escaping the furnace top. As shown in Figure 3, the SB furnace and the two short rotary furnaces located in Building #3 all discharge to a common stack. The rotary furnaces were not operating during the SB furnace test period.

Raw materials are stored in concrete bins in Building #2 in Figure 3. Building #2 also contains the charge bedding area. Building #2 is not large enough to contain all of the raw materials because of the irregular receipt of scrap material. As a result, several large piles of plates, unbroken batteries, and clean lead scrap are stored to the west of Buildings #1 and #2.

The small sodium carbonate water treatment plant is located between the old and new smelters to the south of Building #1. As can be seen in Figure 3, the smelter fence is 200 meters east-southeast from the SB furnace building. This is the closest point from smelter to fenceline. In addition, the prevailing winds are from either the west or west-northwest. However, no ambient measurements were made at the fenceline because of the close proximity of the rotary department (Building #3), the old smelter, and the motorway which lies approximately 200 meters from the fenceline.

Summary of Results

The purpose of the testing performed for this program was to characterize the operation of and emissions from the combined SB smelting and flue dust agglomeration furnaces. The EPA characterization included: 1) an analysis of the smelter feed materials, 2) a determination of the flows of lead, antimony, arsenic, chlorine, and sulfur, 3) measurement of stack emissions, 4) calculation of an emission factor, 5) measurement of lead-in-air levels in the smelter yard, and 6) a description of the furnace operation. The NIOSH characterization included 1) workplace and personal monitoring, 2) evaluation of ventilation and other control systems, and 3) observation and evaluation of work practices and personal protective equipment.

The major results are:

- The smelter feed consisted of the following materials:

Whole Batteries	
-polypropylene case	12.6
-hard rubber case	12.6
Battery Plates	31.5
Agglomerated Dust	3.2
Drosses	3.2
Return Slag	22.1
Coke	5.7
Scrap Iron	1.9
Mill Scale (FeO)	6.3
$CaCO_3$	0.9
Total	100.0

The total input materials contained an average of 79 metric tons lead, 1.5 metric tons antimony, 29 kilograms arsenic, 0.1 metric tons chlorine, and 2.6 metric tons sulfur per day.

- During the test period, these five elements were distributed in the smelter output materials as shown in Table 2.

Table 2. APPROXIMATE ELEMENTAL DISTRIBUTION IN SMELTER EXIT STREAMS

	Percent of total incoming flow				
Output stream	Pb	Sb	As	Cl	S
Lead Bullion	87.7	98.5	10.4	12.8	0.14
Lead stone (matte)	0.90	11.2	75.7	36.8	98.0
Stack gas	0.0025	0.84	0.07	75.8	7.2
Total	89	110	86	125	105

(Estimated accuracy of elemental flow rates is ±20%.)

- The chlorine content of the agglomerated flue dust averaged 25 percent by weight. This indicates that chlorine accumulates in the flue dust collection system.

- The lead particulate emission rates based on the EPA Method 5 test results were:

  -September 26, 1978 0.056 kg Pb/hr
  -September 27, 1978 0.074 kg Pb/hr

- The stack lead particulate emission factors based on the EPA Method 5 test results were:

  -September 26, 1978 0.019 kg Pb/metric ton Pb product
  -September 27, 1978 0.016 kg Pb/metric ton Pb product

- Total stack lead emissions, based on the Radian wet electrostatic precipitator (WEP) experiments, ranged between 0.042 and 0.12 kg Pb/hr.

- Stack chlorine emissions ranged from 1.6 to 7.1 kg/hr.

- Stack gas concentrations ranged from 39 to 54 ppm sulfur during the test. This corresponds to sulfur emission rates of 6.7 and 9.1 kg S/hr.

- Total stack antimony emissions ranged between 0.52 and 0.54 kg Sb/hr.

- Total stack arsenic emissions ranged between 0.0005 and 0.0013 kg As/hr.

- Only one high volume area sampler was run on each of the two test days. The hi-vol was placed underneath the baghouse nearest the east fenceline (see Figure 3). It was approximately four meters from the east agglomeration furnace and 80 meters from the east fenceline. The two twenty-four hours average lead-in-air levels were:

  -September 26, 1978 12 μg Pb/m^3
  -September 27, 1978 18 μg Pb/m^3

- Stack emissions based on the WEP tests are presented in Table 3:

Table 3. STACK EMISSIONS DETERMINED USING WEP TRAIN

	September 26, 1978		September 27, 1978	
Element	Concentration (μg/Nm^3)	Emission rate (kg/hr)	Concentration (μg/Nm^3)	Emission rate (kg/hr)
Lead	1010	0.12	350	0.04
Antimony	4390	0.54	4370	0.52
Arsenic	4	0.0005	11	0.0013
Chlorine	13300	1.6	59500	7.1
Sulfur	55700	6.7	77100	9.1

- The engineering and work practice controls of employee exposure at this smelter are exemplary. The effectiveness of this system of controls is evidenced by the control of employee exposures to lead in all work activities associated with the SB furnace to approximately 100 μg/m^3 or less.

- In general, the local exhaust ventilation systems provided for the SB furnace are well designed and maintained. They provide good enclosure of emission sources, vigorous hood face and duct transport velocities, access openings and mobility to allow efficient performance of routine work.

- Yard sprinkling and washdown procedures appeared to greatly minimize entrainment of dust into the air, tracking muddy materials into other work areas, and splashing of mud on employee clothing and plant equipment.

- Washdown procedures employed in the SB furnace operating area did maintain floors in dust free and clean conditions.

- Reported blood lead data ranged from less than 30 to 80 µg Pb per 100 ml of whole blood.

- Respirators are not worn at the Bergsøe smelter except for unusually hazardous jobs such as furnace maintenance.

- The SB furnace process control room was found to be contaminated with lead (38 to 54 µg Pb/m^3). Some improvement could be made in this area by supplying fresh or filtered air to the control room.

The specific measurements and complete results are available from either the EPA or NIOSH offices in Cincinnati as mentioned earlier.

RESULTS OF THE INITIAL DEMONSTRATION SITE CHARACTERIZATION

At present, only a portion of the results for the "before" or preliminary characterization are available. The preliminary results indicate that lead-in-air levels at the demonstration site are significantly higher than those measured at the Bergsøe smelter. When the complete results are available, an interim report will be issued with a complete list of recommended controls and BMP's intended to reduce the lead-in-air to acceptable levels. Some preliminary results are presented below.

Smelter Description

The secondary smelter demonstration site is typical of existing facilities in the United States. Scrap batteries are broken using slow-moving shears. The battery plates, mud, plant scrap from an adjacent battery manufacturing facility, drosses, and a variety of other lead scrap are fed to a vertical shaft blast furnace. This blast furnace is charged using a skip hoist and traditional slag-tapping and metal-tapping equipment is employed.

The smelter employs approximately 12 people per shift. Much of the work is manual. Charging the furnace and handling raw and refined lead in the smelting building involves simple materials handling equipment. A small Bobcat loader is utilized to handle charge materials within the smelting building.

The building which houses the blast furnace, refining kettles and other smelting equipment is of open-air construction (refer to Figure 4). Several of the walls are open to allow movement of fresh air in and out of the building. The floor of the building is paved but rough. It is heavily contaminated with muds formed by water and paste from battery plates, as well as other materials tucked in from the yard surrounding the smelting building. Housekeeping at this particular smelter is a difficult problem due to the rough surface of

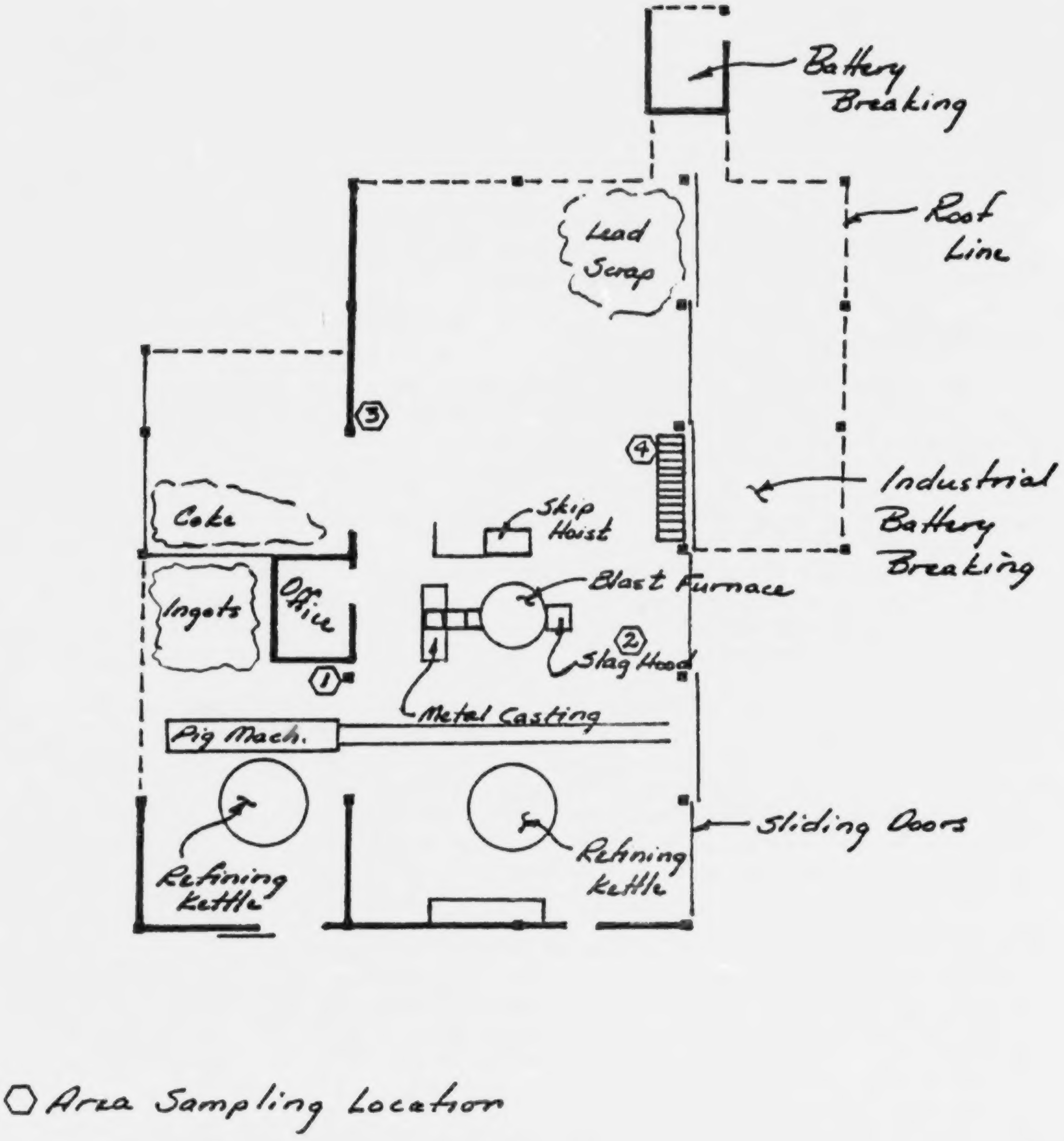
Battery Breaking
Roof Line
Lead Scrap
3
4
Industrial Battery Breaking
Coke
Skip Hoist
Ingots
Office
Blast Furnace
2
Slag Hood
1
Metal Casting
Pig Mach.
Sliding Doors
Refining Kettle
Refining Kettle
○ Area Sampling Location

Figure 4. Work area sampling locations within smelter building.

the floor and the lack of appropriate drains for flushing accumulated lead particulate materials.

Local exhaust ventilation is provided for various points of emission associated with the blast furnace and, to some extent, for the refining kettles. The skip hoist shaft, the charging point at the top of the blast furnace, the molten lead tapping port, the molten lead launder and molding area, and the slag tapping port on the blast furnace all receive some local exhaust ventilation coverage at present. The two refining kettles receive minimal local exhaust ventilation. Only one of these kettles is provided with back draft slot hoods to remove some heat and a portion of the emissions from this source. The other kettle is not presently equipped with local exhaust ventilation. The pigging machine is used to form ingots of finished lead and is not local exhaust ventilated.

Employee Exposures

The general conditions described at this smelter are similar to many other secondary smelters. The employee exposures which result from relatively unsophisticated materials handling procedures and marginally controlled emission sources are high by comparison to the recommended limits for employee exposure. The demonstration site smelter, at present, has a very severe employee lead exposure problem. Air sampling data provided by the smelter indicate that exposures have varied over a wide range and incorporated the usual seasonal and process fluctuations. Exposures monitored during the initial part of this survey confirm those sampling results and indicate that the smelter does operate with workroom lead-in-air concentrations well above, or many times the currently accepted exposure limit (refer to Tables 4 and 5).

Breathing zone and work area sampling for lead and antimony was performed on January 23 and 24, 1979. January 23, 1979 was a relatively normal operating day at the smelting complex. The weather was sunny and cold with light winds blowing from several compass directions. Employees at the smelter performed normal tasks and rotated jobs at the halfway point in the work shift as is indicated by the change in employee initials associated with each of the jobs or operations shown in Table 4. January 24, 1979 was a relatively unusual day at the smelting complex. Heavy rains and strong winds were present throughout the period monitored. The strong gusty winds caused vigorous drafts through the smelting building which disturbed the capture characteristics of local exhaust ventilation hoods and caused the introduction of dust into the atmosphere from the building surfaces. By the afternoon of January 24, large volumes of runoff water were entering the smelting complex and leaving on the down hill side. The smelting building itself was very wet with pools of water in most of the work areas.

The sampling conducted on these two days indicates that exposures were generally higher on January 24, 1979. This result is possible due to the strong air currents which entrained contaminants from the capture zone of the local exhaust ventilation hoods. As stated earlier, the smelting work area was very wet. However, fume produced by slag tapping, metal tapping

TABLE 4. RESULTS OF BREATHING ZONE SAMPLING FOR LEAD AND ANTIMONY

Job/operation		1-23-79			1-24-79		
		Sampling interval	Pb* mg/m^3	Sb** mg/m^3	Sampling interval	Pb* mg/m^3	Sb** mg/m^3
Furnace:	(LW)/(DH)	0758-1232	0.51	0.051	0803-1212	0.62	<0.030
	(NH)/(LW)	1300-1547	0.38	<0.044	1234-1559	0.87	0.047
Hoist:	(DH)/(LW)	0803-1205	0.37	<0.031	0801-1233	0.87	<0.027
	(RH)/(JM)	1300-1547	0.20	<0.044	1238-1558	7.8	<0.037
Payloader:	(NH)/(RH)	0807-1300	0.78	<0.025	0807-1300	1.2	<0.025
	(LW)/(NH)	1258-1547	0.48	<0.044	1343-1557	0.83	<0.025
Coke:	(RH)/(NH)	0828-1233	0.18	<0.030	0818-1342	0.66	<0.023
	(DH)/-	1205-1547	0.25	<0.033	---	--	--
Lead Pot:	(WE)/-	0807-1206	0.21	<0.031	---	--	--
	(WE)/-	1206-1425	0.37	<0.064	---	--	--
Foreman:	(CS)/(CS)	0825-1207	0.27	<0.033	0821-1205	0.66	<0.033
	(CS)/(DH)	1207-1547	0.28	0.037	1214-1603	1.2	<0.032
Industrial Battery Breaker:							
	(JM)/(JM)	0805-1300	0.43	<0.025	0815-1237	0.73	<0.028
	(JM)/-	1300-1615	0.35	<0.029	---	--	--
Yardmen:	(DS)/-	0817-1235	0.69	<0.029			
	(DS)/-	1235-1615	0.25	<0.034			
	(RM)/-	0818-1236	0.37	<0.029			
	(RM)/-	1236-1615	0.23	<0.034			
Battery Breaking:							
	(KA)/(KA)	0833-1209	2.5	<0.034	0823-1200	0.26	<0.034
	(WD)/-	0826-1211	3.5	<0.029	---	--	--
	(KA)/-	1210-1515	0.18	<0.040	---	--	--
	- /(WD)	---	--	--	1203-1520	0.20	<0.037

*Time-weighted-average lead-in-air concentration for the period sampled.
**Time-weighted-average antimony-in-air concentration for the period sampled.

and refining operations was still present. On both days, the amount of antimony in the air was very low by comparison to the permissible exposure limit of 0.5 mg/m^3 (29 CFR 1910.1000).

Several employees were found to have extremely high exposures to lead during portions of work shifts on each day sampled. Battery breakers on the first day sampled showed high exposures to lead and the hoist operator encountered very high lead exposure during the afternoon of the second day sampled. In the case of battery breaking, a splash of lead laden solution may have caused excessive contamination. In the case of the hoist operator, the handling of flue dust as it is charged to the hoist could have caused the deposition of a large lead particle onto the filter.

The work area sampling results shown in Table 5 indicated the same trend from the first to second day of sampling. Sampling results of the second day are generally higher than those of the first. Work area concentrations tend to be much lower than breathing zone concentrations and reflect the fact that employees are much closer to the source of emission within the smelting building than were the stationary area samplers positioned at the locations shown in Figure 4.

TABLE 5. RESULTS OF WORK AREA MONITORING

	Sampling Interval	Pb* mg/m^3	Sb** mg/m^3	Sampling Interval	Pb* mg/m^3	Sb** mg/m^3
Employee Breakroom/ Lunchroom	0838-1214 1215-1559	0.15 0.17	<0.034 <0.033	0739-1145 ---	0.11 --	<0.030 --
Location No. 1 (Refer to Figure 4)	0852-1219	0.050	<0.034	0745-1144	0.37	<0.031
Location No. 2 (Refer to Figure 4)	0854-1222	0.089	<0.036	0755-1142	0.33	<0.033
Location No. 3 (Refer to Figure 4)	0910-1224	0.18	<0.038	0749-1144	0.25	<0.031
Location No. 4 (Refer to Figure 4)	0901-1222	0.12	<0.037	0758-1158	0.30	<0.030

*Time-weighted-average lead-in-air concentration for the period sampled.

PLANNED AND PROPOSED CONTROL MODIFICATIONS

Current Control Modifications

The demonstration site smelter is in the process of making major modifications to its smelting complex to better control ambient and workplace emissions. An important element in this modernization program is the Bergsøe Flash Agglomeration furnace. The furnace will work in concert with a new baghouse facility. The new baghouse system is of much greater capacity than the system which is currently in operation. The new baghouse exhaust ventilation system will provide the capability to exhaust large volumes of air from the smelter building. This exhausted air volume will be gathered from strategic exhaust ventilation hoods which will serve major sources of emission associated with the smelting facility. The following list of major emissions sources will be served by the combined old and new ventilation system:

- The charging location at the top of the blast furnace.
- The skip hoist charging elevator.
- The lead tapping and pouring station.
- The slag-tapping hood.
- The refining kettles.

The final hood designs for local exhaust ventilation of these emission sources are in the development stage at present. The first step in implementing the new exhaust ventilation control program will be to attach existing exhaust ventilation hood structures to the new system. The next stage in implementation of the total control program will be to complete the installation of the flash agglomeration furnace. Once the flash agglomeration furnace is operating in concert with the dust collecting system, attention will be given to the final design of local exhaust ventilation hoods for refining kettles, etc. Completion of this control program will take several months. Base-line exposures data were collected while the existing exhaust ventilation system was in operation.

Optimum control of employee exposures and emissions to the ambient environment will require more than simply increasing the amount of air exhausted from the smelting building and improving local exhaust hood designs. The smelter has plans to improve paving of outside yard surfaces to enclose portions of the smelting building (cutting down cross drafts which would interfere with contaminant capture), and to make materials handling modifications which should help to control escape of contaminants into the work environment.

Control Considerations

Through experience with many U.S. secondary lead smelters and in particular the Bergsøe smelting complex in Glostrup, Denmark, it has been learned that very rigorous attention to cleanliness and housekeeping must be observed if a workplace lead-in-air concentration of 200 $\mu g/m^3$ or less is to be met. To approach 200 $\mu g/m^3$, the demonstration smelter must place much more attention on improved materials handling and housekeeping within the smelter building.

There are a number of emission sources which contribute to workroom lead in air contamination at this smelter. Among these emission sources are the following:

- Pulverization of raw materials of lead particulate by materials handling equipment (front end loaders, large and small)
- Manual shoveling and dry sweeping of lead bearing materials, especially settled particulate, flue dust and plant scrap
- Charging of lead bearing materials to the skip hoist
- Emissions from the charging port at the top of the blast furnace
- Emissions escaping capture during process upset conditions such as blow holes
- Emissions from the slag tapping port during slag tapping
- Emissions from tuyere covers during tuyere punching
- Emissions from the lead well, launder, and molten lead molding operation
- Emissions from cooling finished metal molds
- Emissions from floor surfaces disturbed by personnel and machinery traffic
- Emissions from the melting of lead in refining kettles
- Emissions from drossing and skimming operations in refining kettles
- Emissions from the pigging machine as ingots are formed
- Emissions from vibrating surfaces in the roof structure of the building

- Wind blown particulate from outside yard areas, piles of old flue dust, and materials handling operations

- Emissions from particulate which settles on work clothing and is introduced into the workers breathing zone

- Emissions from splashes of lead bearing solution which attach to work surfaces, dry, and then are dislodged by movement of air or rubbing off by equipment or personnel

- Emissions from tote bins used to collect large particles from knockout boxes or at the base of dust collectors

Each of the above emission sources and possibly others will contribute in varying degrees to employee exposures at this smelting facility. To effectively control exposures attention must be given to each of the possible emission sources listed.

Several general principles in secondary smelter design and operation could be applied at the smelter to improve workplace lead-in-air concentrations. These principles are basic to industrial hygiene and are the foundation of what EPA terms "best management practices" (BMP's). These principles are:

- Engineering Control

  - Isolation of process or employee
  - Enclosure of process or employee
  - Ventilation
    - local exhaust (emission control at emission source)
    - dilution of contaminant with fresh or less contaminated air
  - Substitution of existing materials, processes or equipment with less hazardous materials, processes or equipment
  - Modification of the physical state of the source or contaminant, e.g., wetting dry materials
  - Denying emission energy to the source, e.g., not heating metals to unnecessarily high temperatures

- Administrative Control

  - Operations practices
  - Work practices of employees
  - Scheduling of work to minimize exposures during emission-producing operations
  - Rotation of employees to lower exposure areas
  - Training of employees
  - Housekeeping practices, policies, and procedures (although housekeeping equipment such as a central vacuum cleaner would be classed an engineering control)
  - Maintenance practices and policies

- Personnel Protective Equipment Control
    - Respiratory protection
    - Skin protection

Engineering Controls--

A major improvement could be made by separating the raw materials handling and furnace charging operation from the furnace tapping and kettle refining area of the smelter. This could be done by constructing a partition between the bucket elevator and the blast furnace. In this manner, contamination generated by the charging of scrap lead materials would not communicate directly with the furnace operating and refining kettle portion of the smelter building. This technique has been effectively demonstrated at the Paul Bergsøe and Sons secondary smelter in Glostrup, Denmark.

Improvement in the furnace charging operation could be made by improved materials handling techniques. Scrap battery materials and other furnace charging materials could be conveyed directly to the skip hoist by belt conveyor or other means. This could reduce the need for manual or payloader handling of materials. The charge materials storage portion of the building could be better enclosed to reduce cross drafts and entrainment of particulate. A sprinkling system could be installed to wet down dry charge materials. Should the bucket loader still be required, consideration should be given to equipping this vehicle with a filtered-air, enclosed cab which would offer some reduction in employee exposure.

The floors in the charge preparation and handling area and in the smelting and refining area of the buildings should be modified to permit effective cleaning. Modifications should include provision for wet washing to sumps or drains which can be performed routinely and periodically during the workday. These washdown procedures could be replaced by a central vacuuming system. However, the labor and capital investment involved with the vacuum system may be prohibitive. In either case, dry sweeping of lead particulate materials and other waste should be prohibited.

The existing exhaust ventilation hoods serving the blast furnace and refining kettles will be improved. It will be very important to the eventual success of this control application to provide maximum capture of lead contamination at each emission source. The slag-tapping hood, in conjunction with the work practices utilized to tap slag, does appear to do an effective job of controlling emissions. The hoods associated with molten lead tapping can be improved to provide better capture of emission from the lead well, launder and pouring stations. Improved designs for control of this emission source will be developed in the weeks to come.

The emissions associated with furnace charging will be much better controlled by the new exhaust ventilation service. A much increased volume of air will be withdrawn from the charging hood to insure that furnace upset conditions will not create catastrophic release of lead into the work environment. This charging hood will work in concert with improved ventilation of the skip hoist operation.

Attention must be given to the emissions generated in the loading of the skip hoist itself. Existing ventilation is not adequate to capture contamination in this area. Attention should be placed to providing better capture through use of improved hood design along with the increased amount of exhaust volume which will be afforded by the new system. Improved designs will also be developed for refining kettle hoods which will provide control of melting, drossing and skimming.

Administrative Control Programs--

While engineering controls are being designed and installed, it is imperative that employees be afforded maximum protection from exposure through use of respiratory protective devices and rigorous personal hygiene. The present locker room, showering and lunchroom facility was found to be significantly contaminated with lead. This condition offers the distinct possibility for employees to eat food and use smoking and chewing materials which are contaminated by lead.

The contamination of the lunchroom area may result from lead from a variety of sources. It is adjacent to an industrial battery department which may give rise to infiltration of lead contaminated air. A more obvious source of contamination is the traffic of employees dressed in work clothing to and from the lunchroom. Lead particulate lodged on clothing, hardhats, etc. may be dislodged while the materials are removed or during normal movement within the lunchroom.

To solve lunchroom contamination problems will require the redesign of the shower, locker room and lunchroom facility. It is understood that a new building for industrial battery production is being constructed which will make space available adjacent to the existing hygiene facility. This space could be used for expansion and rearrangement of the facility.

Another means of reducing employee exposures during the installation of engineering controls would be to rotate workers to a lower exposure area during a portion of their workshift. It is not known whether this form of administrative control would be feasible since it would require the training of many new operators to serve in the rotating work assignments.

SUMMARY

The results of the sampling study at the Bergsøe smelter indicate that the combination of control technology and BMP's used there for workplace/fugitive emission control is exemplary. The lead-in-air levels measured both inside and outside the smelter are as low or lower than any similar measurements reported for U.S. smelters. Reported blood/lead levels for the Bergsøe smelter workers range from less than 30 to 80 μg per 100 ml of whole blood. These values are comparable to blood lead levels found at U.S. smelters, however, respirators are not worn at the Bergsøe smelter except for unusually hazardous jobs such as furnace maintenance.

The preliminary results from the U.S. smelter demonstration site indicate that lead-in-air levels are extremely high, both inside and outside the smelter. It is hoped that with the new ventilation system, the Bergsøe agglomeration furnace, the institution of BMP's and a general plant cleanup, lead-in-air levels can be reduced to within legal limits. Of additional importance will be the documentation of both the cost of these modifications and any observed drop in blood lead levels in the smelter workmen.

GETTING READY FOR INHALABLE PARTICLES

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ABSTRACT

In support of a reassessment of the total suspended particulate standard now underway by EPA's Office of Air Quality Planning and Standards (OAQPS), three laboratories in EPA's Office of Research and Development (ORD--Health Effects Research Laboratory (HERL-RTP), Environmental Sciences Research Laboratory (ESRL), and Industrial Environmental Research Laboratory (IERL-RTP)--are examining potential measurement requirements. HERL has recommended a 15 μm upper cut size and a second division at 2.5 μm. Current particulate sampling techniques do not provide such data for either ambient or stack. A recommended measurement program has been developed with short-term modifications for existing samples and a longer effort to fully investigate the requirements for more information including stack condensables.

INTRODUCTION

By December 31, 1980, amendments to the Clean Air Act(1) require EPA to update all the air quality criteria documents, including that for particulate matter, and decide whether to reaffirm existing standards or to issue revised standards. Three of EPA's ORD laboratories--HERL-RTP, ESRL, and IERL-RTP, all at Research Triangle Park, NC--are supporting the effort of the Office of Air and Waste Management's (OAWM) OAQPS to reassess the particulate standard.

HERL-RTP examined the data available relating deposition in the human respiratory system to particle size. The summary of their recommendation provides the guidance needed for measurement requirements to support a revised standard:

> "It is recommended that research to develop information for a size-specific standard should focus on 'inhalable particulate' (IP) matter defined as airborne particles $\leq$ 15 μm

aerodynamic equivalent diameter. This particle size range relates to that fraction of particulate matter which can primarily deposit in the conducting airways and the gas-exchange areas of the human respiratory system during mouth breathing. It is also recommended that a second particle size cut-point of $\leq$ 2.5 μm diameter be incorporated in the air sampling devices, based upon considerations of the chemical composition and the size distribution of airborne particles, and on the predominant penetration of particles $\leq$ 2.5 μm diameter into the gas-exchange region of the respiratory tract. Data collected in this size range could be used in conjunction with epidemiological health parameters to refine an inhalable particulate standard in the future."(2)

DEVELOPING A SOURCE MEASUREMENTS PROGRAM

Though the standards to be revised are ambient, the modifications will directly impact upon source measurements. Both source and ambient measurements must measure the same thing in order to evaluate the impact of a source on the regional air quality. A Task Force (headed by James Abbott, IERL-RTP) was formed by ORD to support OAQP's need for emission data while developing a standard. A subcommittee, consisting of John Nader of ESRL and Bruce Harris of IERL-RTP, was appointed to define the source measurement requirements.

EPA personnel involved in the inhalable particulate (IP) program met to develop specifications and priorities to obtain the IP data. The needs and priorities agreed to at that meeting are presented in Table 1.(3) The measurements subcommittee felt that a research program designed to respond to these needs would be extensive and should be formulated with the best available knowledge. A number of prominent aerosol scientists in the U.S. were invited to participate in a workshop to lay the groundwork for such a program.

IP SAMPLER CURVE

The workshop first tackled the development of performance criteria for devices to be used to meet the 15 μm cut point suggested by HERL-RTP. ESRL had already developed a prototype head for ambient samplers and had amassed some calibration data (Fig. 1).(4) This curve approximates the "standard curve" for IP samplers. The workshop attendees felt that these data need to be confirmed and extended by independent researchers. Since it may be some time before the standard curve is accurately known, it was recommended that EPA adopt a standard mathematical performance curve with specified tolerances. Then any newly

TABLE 1. EPA NEEDS AND PRIORITIES FOR EMISSION DATA FROM STATIONARY SOURCES

Priority	Program Element	Description
1	1a	Total mass $\leq$ 15 μm aerodynamic diameter - cut to match ambient monitor - D_{50} = 15 μm ± 2 μm - slope to match ambient monitor
2	1b	Condensable fraction of total mass
	or	
	2c	Discrete size fractions $\leq$ 15 μm
3	2a	Size distribution - continuum $\leq$ 80 μm
4	2b	Size distribution - continuum $\leq$ 15 μm
5	3	Chemical composition of materials in program elements 1 and 2
6	4	Bioassay of materials in program elements 1 and 2

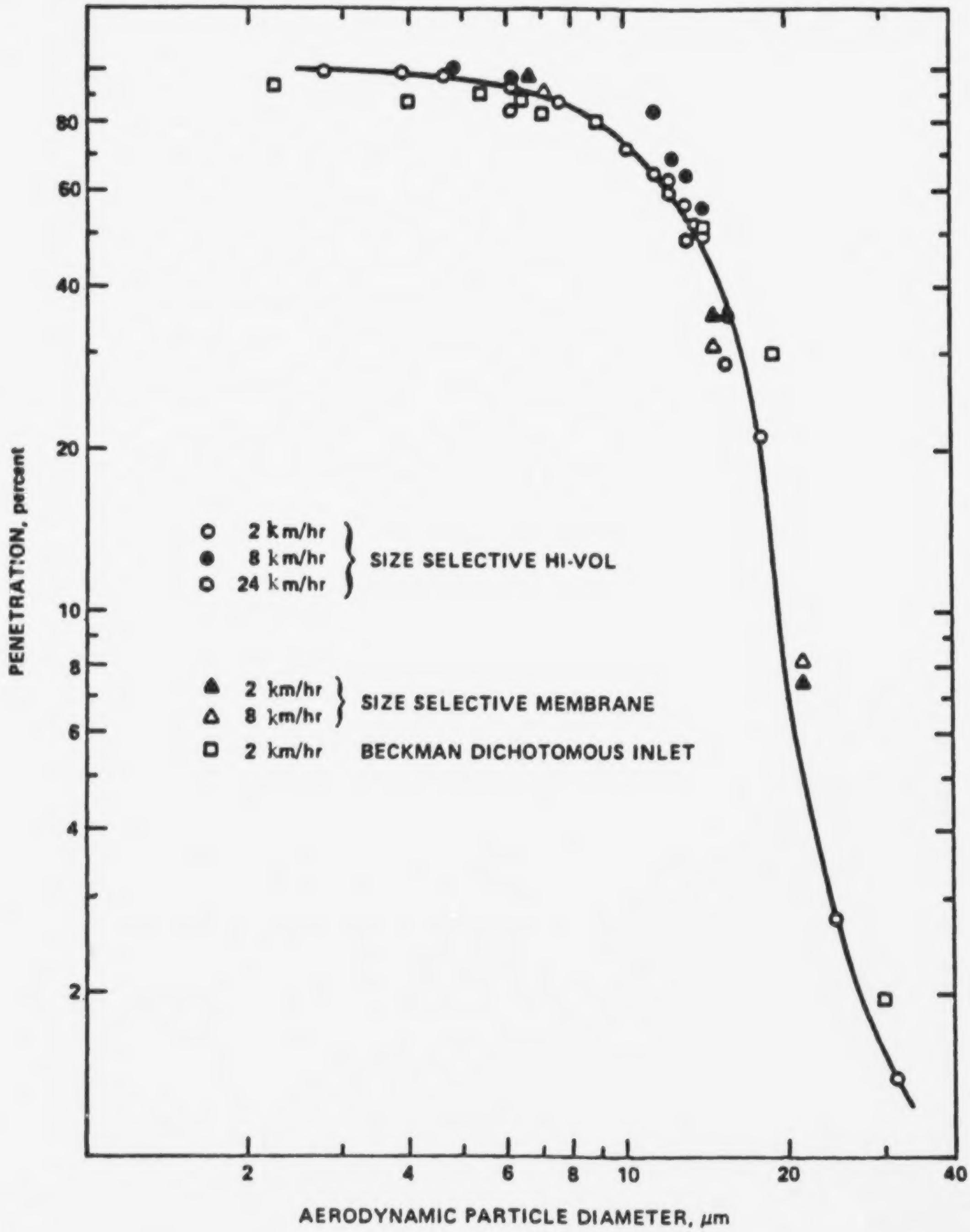

Figure 1. Calibration of the Hi-vol and Dichotomous Sampler Inlets.

developed devices (including the hi-vol and dichotomous sampler) must be calibrated and shown to agree with the mathematical curve within the specified tolerances.

Suggested performance criteria for IP samplers are shown in Fig. 2.(5) All samples would be required to fall within the performance indicated by the shaded area. The shaded area was constructed by plotting two log-normal curves, with geometric standard deviations (σ_g) of 1.3 and 1.7, through the 15 μm, 50 percent point and connecting the 16 percent and 84 percent penetration points. After allowing 10 percent for wall losses of small particles and 10 percent penetration of large particles, the area is completely defined as shown. The tolerance in D_{50} is 15 ± 2 μm. After more thorough characterization of existing and newly developed sampling systems, it may be possible to place more stringent tolerances on a standard curve.

TOTAL INHALABLE MASS SAMPLING

Obtaining total inhalable mass will be markedly different from current practice. Inertial precutters are necessary to meet the proposed performance curve and must be operated at a fixed flow rate to maintain constant performance. Therefore, variation of the flow through the mass sampler, as now done with Federal Reference Method 5 for total mass, cannot be tolerated with existing technology. As a result, the precutters may limit flow rates to a maximum sampling rate of 14 L/M (0.5 ft^3/min) due to the physical size constraints for instack operation.

HERL-RTP indicated the desirability of obtaining information on particles less than 2.5 μm for possible future standards, since particles in this size range are predominately associated with the gas exchange regions of the lungs. Workshop participants felt that this could be met in situ by using a cyclone in series with the 15 μm precutter. This would probably reduce the probe losses reported in many field tests.

The original IP needs list called for the inclusion of condensable material with the total mass. This problem was discussed extensively (presented later in this paper). It was suggested that no measurement system would be sufficiently demonstrated to support a 1980 standard. OAQPS has accepted this but wishes to accelerate development of acceptable methods for condensable material.

PARTICLE SIZING

Instack impactors and series cyclones have been used to obtain extensive particle size data on a number of processes.

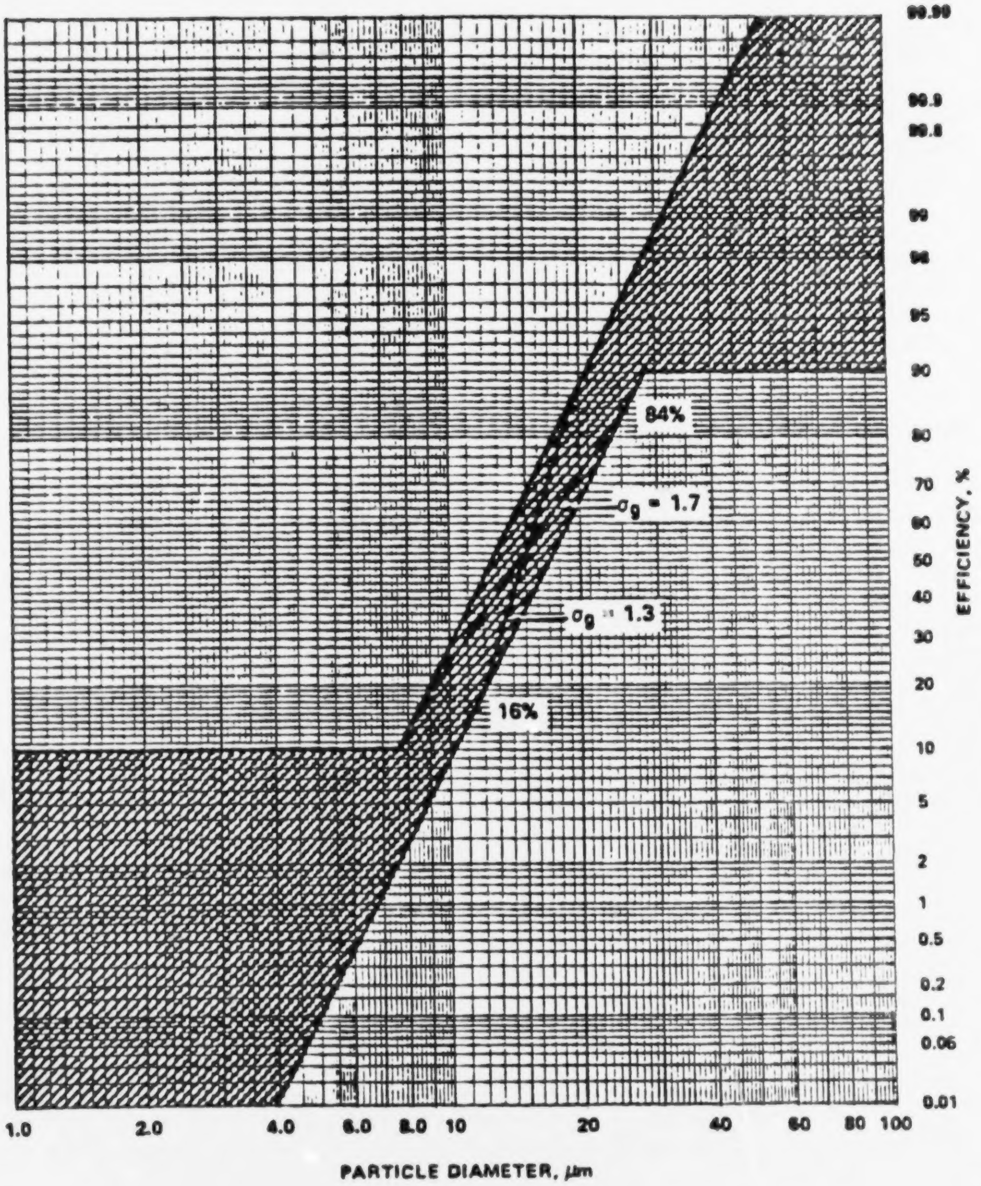

Figure 2. Recommended IP Sampler Specification.

Unfortunately, most current operating practices yield upper cut points of 10 μm or less which do not meet the IP performance needs. Most current inertial sizing devices can meet these criteria by reducing sampling rates. Concern was expressed by the aerosol scientists that the bounce and collection efficiency roll-over observed under current practices may be aggravated by the reduced flow rate. This potential problem and recalibration under reduced flow are currently being investigated.

CONDENSABLE MATERIAL

Measuring accurately the nature of condensable material in a process stream and determining its physical and chemical properties in the atmosphere presents perhaps the most important and challenging task of the IP program. Many process streams contain materials in the vapor state which condense to form homogeneous and/or hetrogeneous aerosol particles. Some "condensable" vapors will not actually appear as particles in the air downwind of the source, since their equilibrium vapor pressure may be high enough that they will not condense, or condensed vapors may re-evaporate as dilution is increased. In order to measure the properties of the aerosol emitted to the atmosphere, the condensation in the sampling system must simulate plume behavior. Lower dilution rates may maximize the supersaturation and overestimate the contribution of condensed vapors to the IP.

The rate of dilution may also be important. In studies of aerosols produced by diesel engines, very rapid dilution of 1000:1 has prevented condensation of organic vapors. Slower dilution may lead to condensation which clearly occurs under some circumstances in plumes near the stack outlet.

The contribution of condensed vapors to mass emissions can be very significant under some circumstances. Consider a coal-fired boiler burning 10 percent ash coal and with 99 percent particle removal. The emitted ash particles will amount to 0.1 percent of the original coal mass. For a conservative estimate of the contribution of sulfates, assume that the coal contains 1 percent S, and that 5 percent of this (or 0.05% of the coal mass) is emitted from the stack as sulfates (primarily sulfuric acid). If all of the sulfate were to condense, it would equal one third the total amount of solid mass emitted.

In other processes, notably smelters, the fraction of mass emitted as vapors and condensing at the stack/ambient interface may far exceed the mass existing in solid form within the stack.

During sampling, it would be preferable to dilute and cool a representative sample of the particulate and gas as it exists in the stack. However, because of the extreme difficulty in transporting large particles through sampling lines and dilution systems without unacceptably high losses, it will almost certainly be necessary to collect and size particles greater than 2-3 μm in the stack without dilution or cooling. The smaller particles would then pass to the out-of-stack dilution system for further analysis. Removing particles from the sample aerosol removes surfaces where condensation likely occurs upon cooling and dilution. Thus, to some extent, the experiment will be biased, and the size distribution and mass concentration of the aerosol in the diluter will be distorted. Experimental and theoretical studies must be performed to determine the magnitude and nature of the errors associated with the compromised procedure.

FUGITIVE EMISSIONS

Nonducted sources are among the most difficult to measure accurately. Current practice is to use ambient sampling techniques in upwind-downwind studies or stack methods for hooded sources. The IP methods developed for the other areas can be incorporated into the existing fugitive methods. Perhaps the greatest need is to develop suitable sampling strategies. Some special research fugitive samplers, such as IERL-RTP's FAST (Fugitive Assessment Sampling Train) and ESRL's Mega-Vol, are being modified to include a 15 μm first cut.

CALIBRATION

An area that cannot be neglected is the calibration of instruments and systems with standard aerosols. The development of calibration procedures is not straightforward and will require considerable time, effort, and expertise. The minimum parameters which must be considered are presented in Table 2. It is important to be able to calibrate devices at more than one facility so that results can be compared and verified.

A RECOMMENDED RESEARCH AND DEVELOPMENT PROGRAM

At the workshop, a program was developed to meet the IP needs outlined at the beginning of this paper. This program is presented in Table 3.(6) Some of the items have already been discussed. Still others represent long-term development which will be undertaken. Many are desirable but, due to funding limitations and low priority, will not be supported by EPA.

TABLE 2. CALIBRATION PARAMETERS

Particle Type - Wet, dry, monodisperse, polydisperse

Velocity of Gas

Particle Diameter - Aerodynamic, standard set for verifying size

Sampling Efficiency vs. Particle Size

Wall Loss vs. Particle Size

Aerosol Concentration - Can affect the operation of some devices

Temperature - Must simulate sampling conditions

Wind Direction - Will certainly have some effect

Loading - Reentrainment

Frequency of Calibration

Number of Devices Calibrated - Initial prototypes, or all?

TABLE 3. TECHNIQUES RECOMMENDED FOR USE IN OBTAINING DATA TO SET IP EMISSION FACTORS FOR INDUSTRIAL SOURCES

Time Scale	Program Element	Stack	Ambient	Fugitive
Immediate	1a. Total Mass ≤ 15 µm			
	1b. Condensable Mass			
	2c. Size Dist. ≤ 15 µm, Discrete			
	2a. Size Dist. ≤ 80 µm			
	2b. Size Dist. ≤ 15 µm Cont.	Re-analyze data		
Six Months	1a. 1b.	Cyclones w/impingers, for 1a, 1b, 2c	Lo-Vol, Hi-Vol, Dichot. (Similar inlets requiring more calibration and performance verification)	
	2c.	Also SASS train	1b, N/A	FAST train w/new inlet
	2a.		Dust fall onto suitable substrate, with automatic or manual image analysis	
	2b.	Cyclones, impactors		
Longer Term	1a. 1b. 2c.	15 µm, 2.5 µm cyclones followed by dilution Run in parallel with Method 17		
Nine months	2a	15 µm, 2.5 µm cyclones Dilution - Out of stack sizing. (imp., opt., diff., elec.), body impactor for 2-80 µm	Rotating, body impactor optical, diffusional, elec., centrifuge Mobile labs	

SUMMARY

An IP standard is being actively considered by EPA. Health considerations suggest an upper size cut at 15 μm, and all sampling methods need to be modified to provide the cut. Programs to provide the information needed to measure IP emissions and to assist OAQPS in reassessing the particulate standard have been developed with assistance from many U.S. aerosol scientists. These programs have been prioritized and many are now underway.

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1. "Amendments to the Clean Air Act of 1970", Public Law 9595, August 1977.

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3. Internal Memorandum, J. S. Nader (EPA/ESRL) and D. B. Harris (EPA/IERL-RTP) to J. H. Abbott (EPA/IERL-RTP), November 1978.

4. Private communication, C. Rhodes (EPA/ESRL) to D. B. Harris (EPA/IERL-RTP), November 1978.

5. "Sampling Techniques for Inhalable Particulate Matter", ed. by W. B. Smith, Special Report for EPA Contract 68-02-2610, January 16, 1979, page 7.

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LEAD AND ZINC INDUSTRY

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AN OVERVIEW OF CONTROLS IN PRIMARY LEAD AND ZINC

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ABSTRACT

The lead-zinc processes are briefly reviewed with emphasis placed on the potential particulate emission points. Particulate emissions from the industry are discussed with an emphasis on specific problems such as lead and trace metals emissions. The use of particulate control equipment is reviewed and a number of outstanding control problems are highlighted. New or emerging process technology is discussed especially in relation to its impact on particulate emissions control. Existing North American environmental legislation and pending legislation will be summarized.

INTRODUCTION

Lead and zinc frequently occur together in the same ore. Consequently the primary outputs of the two metals are often interrelated and mines frequently produce significant tonnages of both. The normal practice is to produce separate lead and zinc concentrates, using froth flotation wherever possible, and treat them in separate smelting and refining processes. In order to provide a suitable grade of lead concentrate and zinc concentrate, it is often necessary to extract a middlings fraction as a lead-zinc concentrate; this can amount to between 10% and 15% of the total concentrate produced. Most of this is treated in the Imperial Smelting Process. Several alternative processes are available to recover the metals and producers select and modify the processes to suit their particular conditions. Important considerations include concentrate composition and environmental and economic aspects.

In 1977 approximately 63 primary lead and 76 primary zinc smelters operated throughout the non-communist world. Tables 1 and 2 show the total rated output capacities according to the principal processing methods.

TABLE 1. ESTIMATED LEAD OUTPUT CAPACITY OF PLANTS IN THE NON-COMMUNIST WORLD ACCORDING TO PRINCIPAL PROCESSING METHODS (AMERICAN BUREAU OF METALS STATISTICS 1977)[1]

Method	No. of Plants	Refined Lead Output STPY
Conventional	54	3,941,000
Imperial Smelting	8	290,000
Boliden Electric Furnace	1	55,000
Total	63	4,286,000

TABLE 2. ESTIMATED ZINC OUTPUT CAPACITY OF PLANTS IN THE NON-COMMUNIST WORLD ACCORDING TO PRINCIPAL PROCESSING METHODS (ABMS, 1977)

Method	No. of Plants	Refined Zinc Output STPY
Electrolytic	44	4,182,000
Pyrometallurgical	24	1,354,000
Imperial Smelting	8	672,000
Total	76	6,208,000

Table 3 summarizes the total world production of lead and zinc, including the Communist Bloc. Note, Tables 1 and 2 refer to the related capacities of the processes and not the actual yearly outputs.

TABLE 3. WORLD PRODUCTION OF REFINED LEAD AND SLAB ZINC FOR 1977 AND 1976 (ABMS, 1977)

Region	Lead Ouptut 1976	1977	Zinc Output 1976	1977
N. America	1,042,472	995,066	1,249,792	1,167,875
S. America	171,800	183,000	157,722	164,580
Europe	993,614	1,045,290	1,752,765	1,847,000
Asia	250,969	257,521	878,383	934,100
Africa	114,700	123,700	204,300	201,800
Australia	235,300	233,100	275,100	282,400
Sub-total	2,808,855	2,837,677	4,518,062	4,607,180
Communist Bloc	1,212,000	1,211,800	1,873,300	1,884,900
Others	16,300	15,900	-	-
Total Production	4,037,155	4,065,377	6,391,362	6,492,080

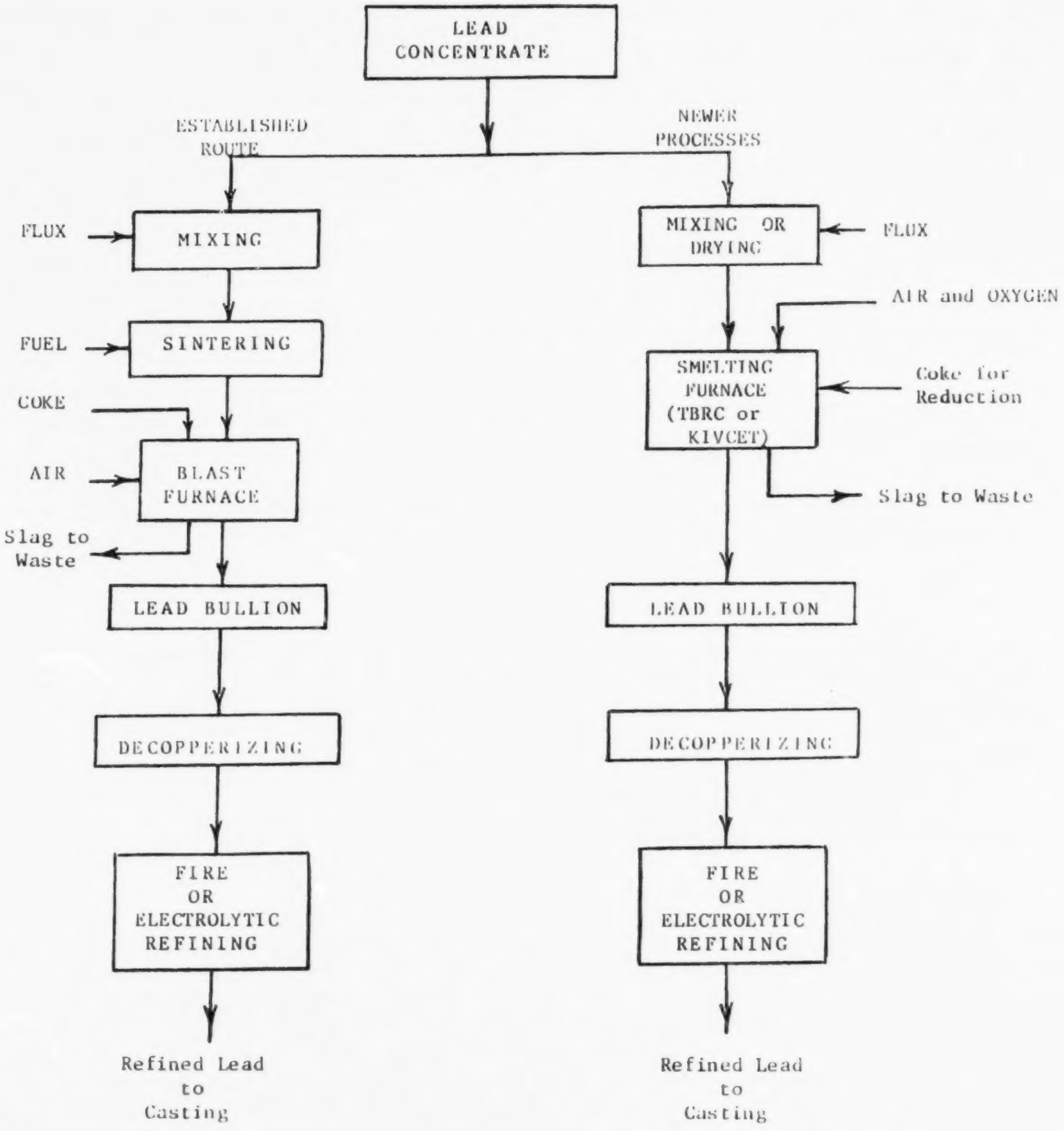
Primary lead and zinc processing plays an important role in an industrial society. The problems faced by lead and zinc producers in controlling particulate emissions, and conforming to occupational hygiene standards are similar to those faced by other sectors of the metallurgical industry[2].

THE PRIMARY LEAD INDUSTRY

Lead Production Processes

Most of the world's primary lead is produced from sulphide concentrates in reduction blast furnaces. Figure 1 shows a generalized flow sheet. The conventional process consists of feed preparation, sintering, blast furnace reduction, bullion drossing and either electrolytic or fire refining. This process accounts for approximately 92% of installed western world capacity. A similar process, the Imperial Smelting process, produces an additional 7% from zinc-lead sinter. Several new processes have been developed that eliminate the need for

FIGURE 1 GENERALIZED LEAD PRODUCTION FLOWSHEET


LEAD CONCENTRATE
ESTABLISHED ROUTE
NEWER PROCESSES
FLUX
MIXING
FUEL
SINTERING
COKE
AIR
BLAST FURNACE
Slag to Waste
LEAD BULLION
DECOPPERIZING
FIRE OR ELECTROLYTIC REFINING
Refined Lead to Casting
MIXING OR DRYING
FLUX
AIR and OXYGEN
SMELTING FURNACE (TBRC or KIVCET)
Coke for Reduction
Slag to Waste
LEAD BULLION
DECOPPERIZING
FIRE OR ELECTROLYTIC REFINING
Refined Lead to Casting

sintering and premit direct smelting of lead concentrates; these offer advantages in environmental control and occupational hygiene. Examples are the Kivcet, the Boliden TBRC and the Boliden electric furnace.

In both the conventional and Imperial processes, concentrates are sintered to produce a hard, agglomerated oxide charge for the blast furnace. The feed is first thoroughly blended in bedding or blending plants and pelletized. A low sulphur content is required, and necessitates the recycling of large amounts of crushed sinter to dilute the fresh concentrate. Sintering involves heating the charge to just below its melting point on a moving grate. Up-draught or down-draught machines are used. Temperatures of about 1000°C are attained at the combustion front. Part of the volatile metals present are evolved, e.g., mercury, arsenic, lead, cadmium and zinc, and are contained in the off gas together with SO_2 and entrained particulates. The sintering machines are normally operated to produce gas with 4-7% SO_2 which can be treated in conventional contact acid plants. The gases from different wind boxes may be split into strong and weak streams and the weak stream (less than 2% SO_2) discharged following particulate recovery. Alternatively, the weak gases may be recirculated and a single, strong off gas stream produced. The bulk of the particulates are recovered in electrostatic precipitators or fabric filters. The acid-plant wet-gas cleaning train provides further control of particulate emissions, by collecting volatile and particulate constituents which escape primary collection, for disposal in the acid plant sludge.

The sinter is crushed, screened and charged with coke and flux to a blast furnace, where it is reduced and melted. Coke reduces lead oxide and impure lead bullion separates from slag in a settler. The furnace gas is drawn from the top and cooled from about 750° to 140°C by the addition of cooler air. Suspended particulates are recovered in fabric filters. The slag is tapped and granulated with water, or delivered molten to a slag dump. Depending on the zinc content, slag may be fumed by the injection of coal in a reverberatory, or an electric furnace, to recover zinc.

The Imperial Smelting Process is similar to the conventional blast furnace and is considered primarily as a zinc process. It is described in this section because of its similarity to the lead blast furnace. The furnace uses preheated sinter, hotter blast air and higher shaft operating temperatures. Zinc is vapourized in the shaft and recovered by shock cooling the gas in a lead splash condenser. A wet scrubber captures residual vapours and particulates escaping the condenser.

The impure lead bullion is cooled in drossing furnaces. Some impurities principally copper and nickel, and to a lesser extent antimony and arsenic, are precipitated according to their solubility limits. Batch or continuous processes are used.

The drossed bullion is refined to produce lead and certain by-products. Fire or electrolytic refining is used. Fire refining involves the sequential removal of dissolved impurities by reacting molten lead with reagents at controlled temperatures. Various drosses are formed and removed mechanically. Electrolytic refining involves casting the bullion as anodes, and plating out lead on cathodes in electrolytic cells using hydrofluosilicic acid. The impurities are retained as a sponge on the spent anodes, and are refined to recover by-products.

Several stages are involved in pyrometallurgical refining and alternative methods can be used for each stage. The stages are:

1. Softening

   Molten lead is oxidized to form a dross which contains the arsenic, antimony and tin. Oxidation may be carried out using an air lance in a reverberatory furnace, or by stirring lead with sodium nitrate and caustic soda in the presence of air in a reaction kettle at 700°C. The drosses are removed and smelted to recover by-products. An alternative method passes molten lead through an immiscible sodium salt melt. The impurities dissolve in the salt melt and are subsequently recovered hydrometallurgically.

2 De-silvering

   Zinc is stirred into batches of softened bullion, or bullion is poured through molten zinc in a continuous desilvering column. The precious metals dissolve in the zinc. Temperatures are controlled so that zinc, containing the precious metals, precipitates from solution to form a crust.

3. De-zincing

   A vacuum is maintained over molten lead at 600°C and zinc is distilled and condensed on water-cooled hoods. Batch or continuous processes are used.

4. De-bismuthing

   Calcium and magnesium alloys are reacted with the lead in processes similar to desilvering. Bismuth dissolves preferentially in the melt and forms a scum. Batch or continuous processes are used.

> The final lead is treated with sodium hydroxide to remove residual impurities and cast as ingots. Gaseous and fume emissions are controlled by ducting the gas to high efficiency control devices. The gases have to be cooled prior to collection from operations where arsenic is evolved.

Electrolytic refining isolates the impurities in the anode slimes and minimizes the amount of material to be treated pyrometallurgically. The slimes are melted, with fluxes, in a small reverberatory furnace. A tin-bearing slag and a metal are formed. The slag is smelted to recover tin. The molten metal is transferred to another furnace and lanced with air. Arsenic and antimony are fumed off, and a lead-bismuth slag and a dore metal formed. Gold and silver are recovered by cupelling and electrolyzing the dore metal. The bismuth slag is smelted and chloridized to recover bismuth. Gases containing arsenic and antimony are generated from the pyrometallurgical process and the gases have to be cooled to condense the vapours. Fine particulates are formed and high-efficiency fabric filters have to be used to control emissions. Arsenic trifluoride may be formed by reaction with residual electrolyte; this has a relatively high vapour pressure at the fabric filter operating temperature, and secondary scrubbing may be required to reduce arsenic and fluoride emissions to acceptable levels.

Alternative processes exist to smelt concentrates directly to lead bullion. Boliden have developed a top blown rotary converter (TBRC) to process high-grade reduction phases; lead is smelted in a 60% oxygen stream and slag and bullion formed; a high-grade SO_2 gas is produced which is treated by waste heat boilers, electrostatic precipitators and sulphur recovery plants. The gas volume per unit of concentrate treated is low by virtue of the oxygen enriched air used. A large proportion of the lead is contained in the slag and is reduced by the addition of coke. The off gas is diverted to a baghouse during the reduction mode. The process offers advantages in environmental control; the unit is relatively small, SO_2 concentrations are high because of the use of oxygen and a secondary hood can be used to capture fugitive emissions.

Boliden also use a combination of an electric furnace and converter to produce lead bullion. Dry concentrate and flux are smelted in the electric furnace freeboard and additional heat supplied by the electrodes. Complete desulphurization is avoided to minimize slag losses; further desulphurization takes place in a converter. Particulates are recovered from the furnace and converter gases. The furnace gas is treated in sulphur recovery plants. The magnitude of the emission control problem is less than that of the conventional sinter plant-blast furnace process.

The Kivcet process has been developed in Russia to smelt lead-zinc concentrates directly in a combination of an oxygen flash and an electric furnace. Concentrates are melted in the oxygen flash shaft furnace and flow under a partition wall into an electric furnace. Coke is added to the slag and zinc fumed off. Lead bullion is tapped from the furnace. Two gas streams are produced, a strong SO_2 flash stream and a weak SO_2 electric furnace stream. The former will be treated in sulphur recovery plants and the latter discharged following zinc recovery. The process has not yet been applied in the Western world. It offers advantages in pollution control but may be limited by capital and operating costs.

Particulate Pollution Problems and Abatement

Lead is a widely dispersed element and is usually present in trace amounts in food, air and water. It serves no useful function in human metabolism and elevated levels cause adverse effects. Lead is a cumulative poison and its concentration in the environment has to be minimized. In the elemental and sulphide forms it is relatively safe from a health point of view but as an oxide it is readily absorbed by the body, particularly if present as fumes smaller than 10 microns, which are absorbed by breathing. The control of particulate and fine oxide fumes generated during smelting and pyrometallurgical refining operations is important. Emissions can be controlled using high-efficiency particulate collection devices.

The emission of other metals may also occur, namely arsenic, cadmium, zinc, antimony and mercury. Measures to abate lead emissions are usually effective in abating emissions of these other elements as well.

Emissions can occur from a number of sources in the plants, including both the process off gas streams, and also the fugitive escape of particulates and fumes to the general work area. The latter include sources such as material storage and handling, crushing and screening, leaks from process equipment, tapping operations, molten metal transfer,etc. Ventilation systems are installed to remove fugitive emissions from the working environment. Proper design and good housekeeping and maintenance practices can limit the severity of fugitive emissions. Two types of gas stream have to be cleaned, process off gas and ventilation-gas. The type of process used affects the magnitude of the control problem.

The sinter plant-blast furnace processes represent the major method of bullion production. Emissions from the sinter-plant process off gas stream can be effectively abated by directing all the process gas to sulphur recovery plants. The bulk of the particulates and condensed fumes are removed in electrostatic precipitators and de-misting towers in the sulphur recovery plants. Contact sulphuric acid plants are the usual abatement process and normally require SO_2 contents greater than 4%. This requires specialized operation of the sinter strands with gas recirculation. In some cases weaker gases from certain wind boxes may be segregated and treated separately, usually being diverted to the blast furnace baghouse. In other cases sulphur is not recovered from the gas, and gas is discharged following particulate recovery. Emissions are higher for these cases.

Some ores contain trace amounts of mercury which raises concern because of its toxicity. Nearly all of the mercury is driven off in sintering and most of it is collected in the wet-gas cleaning section of the sulphur recovery plants, if sulphur abatement is used. Otherwise, most of the mercury will be emitted. The acid plant waste sludge has to be safely disposed of to minimize potential environmental effects. The mercury content of the acid may also have to be reduced to meet acid product specifications. For instance, many acid consumers now specify mercury concentrations below 0.5 ppm in their contracts.

Emissions from blast furnaces are controlled using high-efficiency fabric filters. The gases are cooled prior to collection using air drawn from plant ventilation systems or outside. The former is preferred since it reduces the volume of ventilation gas to be treated in other facilities. Emissions from the Imperial Smelting furnace are controlled by wet scrubbers downstream from the zinc condenser.

Fugitive emissions are usually a major problem for the sinter plant-blast furnace process. Properly designed ventilation systems have to be installed. The newer direct smelting processes, such as the Boliden TBRC and the Kivcet offer advantages in the elimination of sintering and the attendant handling of large amounts of recycle material, coupled with more compact plants, fewer unit-processes and lower off gas flow rates. The volume of exhaust gas to be cleaned is lower and less effort and expense are required to meet occupational hygiene standards.

Particulate, arsenic, and antimony emissions can occur from drossing. Emissions are controlled by cooling the gas to condense the volatile metal vapours and passing the gas through high-efficiency fabric filters to capture the fine particulates. Drossing removes much of the arsenic and reduces the magnitude of potential arsenic emissions in down-stream refining operations. Continuous drossing processes eliminate the fugitive emissions from hot metal transfer.

Lead can be refined electrolytically or by fire refining. The electrolytic process offers advantages in industrial hygiene and pollution control as pyrometallurgical processes are limited to by-product recovery, and intermediate transfers of molten material are absent. Electrolytic refining is used extensively in Japan where strict environmental standards apply. Fire refining is favoured in many instances because of its lower capital and operating costs, and adequate controls can be installed to meet environmental and occupational hygiene standards. The facilities have to be properly designed[3]. The magnitude of particulate emissions and industrial hygiene problems occuring from different stages in fire refining varies and depends on the method used[3]. Emissions can be controlled using properly designed flue and ventilation systems, duct process exhaust and fugitive emissions[4]. High-efficiency collection devices are used. Emissions are most serious from the softening and softening dross treatment processes, where arsenic and antimony vapours are evolved. The gases have to be cooled to condense the vapours and cleaned in high-efficiency fabric filters. The other operations (de-silvering, de-zincing, de-bismuthing and by-product treatment) have to be properly hooded and ventilated. Exhaust gases are cleaned in fabric filters. Some fire refineries are designed so processes operate continuously or semi-continuously. This minimizes fugitive emissions from material transfer.

Slimes treatment, and anode cathode casting, are the only sources of particulate emissions from electrolytic refining. Slimes treatment is the major source of emissions. Large amounts of arsenic and antimony are volatized in the process and the gases have to be cooled, to condense the vapours and the fine particulates captured in high-efficiency fabric filters. Arsenic trifluoride and other fluorine compounds may be formed as a result of reaction with residual electrolyte; significant concentration can be present as vapours even after cooling and particulate recovery, because of the high vapour pressure. Secondary cleaning in wet scrubbers may be used. The problems may be minimized by washing the slimes to remove electrolyte prior to treatment.

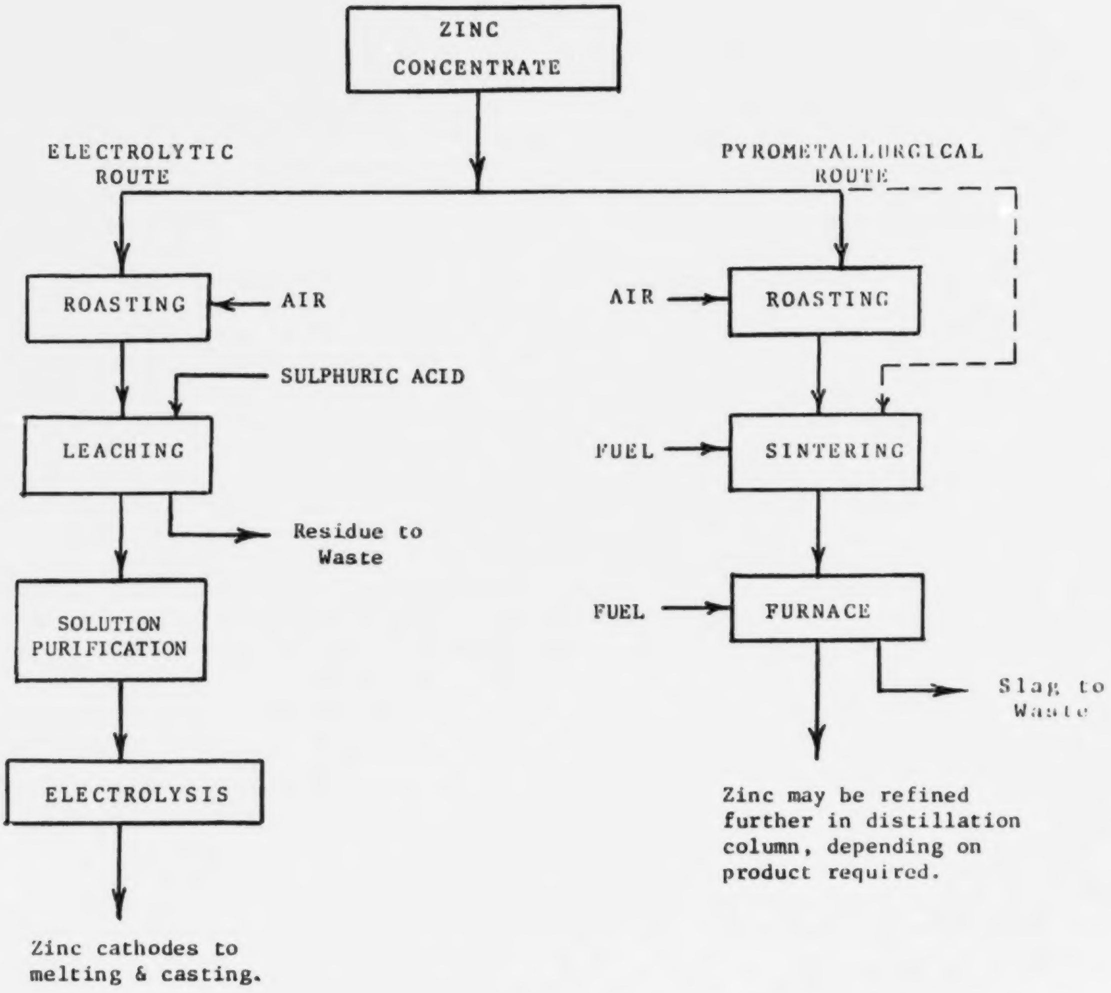
THE PRIMARY ZINC INDUSTRY

Zinc Production Processes

Zinc is produced from sulphide concentrates by processes which include pyrometallurgical operations. Two processes are used: electrolytic (hydrometallurgical) and pyrometallurgical refining. Figure 2 shows a generalized flowsheet. The electrolytic process is dominant and accounts for approximately 67% of the Western world capacity. The pyrometallurgical process includes the Imperial Smelting Process described previously under Lead Production.

Roasting is used in all the electrolytic and most of the pyrometallurgical refineries; some pyrometallurgical refineries sinter the concentrates directly without pre-roasting. Roasting

FIGURE 2 GENERALIZED ZINC PRODUCTION FLOWSHEET


ZINC CONCENTRATE
ELECTROLYTIC ROUTE
PYROMETALLURGICAL ROUTE
ROASTING
AIR
SULPHURIC ACID
LEACHING
Residue to Waste
SOLUTION PURIFICATION
ELECTROLYSIS
Zinc cathodes to melting & casting.
AIR
ROASTING
FUEL
SINTERING
FUEL
FURNACE
Slag to Waste
Zinc may be refined further in distillation column, depending on product required.

converts the zinc to an acid soluble form in electrolytic refining, and a reducible, volatilizable form, in pyrometallurgical refining. Three types of roasters are used: fluid bed, multi-hearth and suspension roasters. The fluid bed is the most recently developed and has been installed in newer facilities and replaced others in many existing plants.

Fluid beds provide fast intimate reactions between the sulphides and air. High temperatures are generated by the autogenous combustion and heat is recovered from the off gas in waste heat boilers. Up to 80% of the product exits in the off gas and is recovered by waste heat boilers, cyclones and electrostatic precipitators. The gas contains 8-13% SO_2 and is treated in sulphuric acid plants. These plants have wet scrubbers, wet electrostatic precipitators and demisting towers ahead of the catalyst, to remove residual particulates to protect the catalyst beds. Particulate emissions are virtually absent, since any material passing the electrostatic precipitator is removed in the acid plant.

In multi-hearth roasters concentrate is roasted as it is contacted by hot combustion gas while being raked down through a series of hearths in a cylindrical vessel. Coal may be added and fuel burners used to supply heat. The reaction is slow and excess air has to be added to obtain the required desulphurization. The SO_2 contents are lower than those of fluid beds and SO_2 fixation is not always possible. Gases with approximately 5% SO_2 can be produced and treated in acid plants, if good seals are maintained. The hearth temperatures vary between 450° and 700°C. Fewer solids are entrained in the off gas than in the case of fluid beds. The multi-hearth roasters have greater flexibility to eliminate volatile elements, such as arsenic and lead, since the hearth temperatures and oxidation conditions can be varied. There is a greater tendency to fix these as nonvolatile oxides in fluid bed roasters. The potential for particulate emissions is greater when acid plants are not used.

Suspension roasters are a hybrid of fluid bed and multi-hearth roasters. The concentrate is dried on the lower drying hearths and then charged to the upper combustion chamber. Approximately 60% of the product is entrained in the off gas and recovered in waste heat boilers, cyclones and electrostatic precipitators. The balance comes from the lower roasting hearths. Gas containing 10-13% SO_2 is produced and treated in acid plants.

The roasted product is leached and zinc recovered electrolytically, or sintered and treated in pyrometallurgical volatilization processes. Electrolytic recovery results in only minor particulate emissions; however, acid mist is generated and good ventilation has to be provided. The calcine is leached in sulphuric acid and zinc dissolved. The solution is purified in a number of stages and various by-products are produced. Iron is precipitated by adding reagents, and copper, nickel, cadmium and cobalt are precipitated by adding zinc and arsenic trioxide dust. The purified solution is processed in electrolytic cells and zinc is plated on inert aluminum cathodes. Sulphuric acid is regenerated and recirculated to the leach process. The cathodes are periodically stripped, and the zinc sheets melted and cast as slabs or other shapes.

In the pyrometallurgical process the roaster product is sintered to volatilize and recover lead and cadmium, and produce a suitable fused, porous feed for the retorts. Down-draft sinter machines similar to those used in lead practice are employed. Some plants sinter mixes of fresh concentrate and recycle sinter on the grates without the use of roasters. Different quality feeds are required for the three retort processes and the sintering practices differ.

Pyrometallurgical zinc volatilization processes utilize the relatively low boiling point of zinc, 906°C, to volatilize zinc from a reduced mixture of sinter and coke. The zinc vapour is shock cooled to prevent reoxidation. Three types of retort are used; horizontal, vertical and electrothermic. The Imperial Smelting Process relies on the same principles. The retort product may be marketed directly or refined further in distillation columns.

Horizontal retorts are the oldest reduction system presently in use. Many of the plants have been phased out, particularly in North America and Europe owing to the high cost of labour and the inability to control emissions. The plants are only viable in areas where labour costs are low. The furnace consists of a number of cylindrical retorts packed in a slotted shell. The tubes contain a mixture of sinter and coke. Hot combustion gas is passed through the furnace around the tubes and zinc volatilized and condensed in a clay outer wall condenser. The operation is a batch process requiring about 48 hours. The process is not very attractive because of excessive operating costs, low productivity, high labour demands for tedious tasks (e.g., packing and stacking tubes), difficulties in controlling particulate emissions from the numerous retorts, and low zinc recoveries.

The vertical retort was developed by New Jersey Zinc in the 1920's. The process is semi-continuous. Hard or briquetted sinter is charged to the shaft in slugs and moves downwards. The charge is heated indirectly through an outer shell and zinc volatilized and shock cooled in a splash condenser. The condenser gas is cleaned in a wet scrubber and injected into the fuel burner. Particulate emissions can be minimal with good operating practices.

The electrothermic process was developed by St. Joe Mineral Corporation in 1936 and is in operation at Monaca (U.S.A.) and Toyama (Japan). Preheated, hard sinter and coke are charged and heated by electric arcs between electrodes in the upper and lower shaft. Zinc oxide is reduced and zinc volatilized. Gases are drawn off midway from the furnace and passed through a condenser full of liquid zinc. The exhaust is cleaned in a wet scrubber and the carbon monoxide containing gas used as auxiliary fuel.

The zinc products from the different retorts may be marketed directly or refined in vertical fractional distillation columns to produce zinc to meet different specifications. Principal elements removed are cadmium and lead.

Particulate Pollution Problems and Abatement Technology

Particulate and metal vapour emissions can occur from zinc production. Emissions from electrolytic refining are more readily controlled since only one pyrometallurgical process (roasting) is involved. Potential emissions from the processes include suspended particulates, and lead, zinc, cadmium, arsenic mercury, etc., which may be emitted as vapours or fine condensed fumes. High-efficiency control devices are required.

The roaster emissions are effectively abated when SO_2 recovery is used. This is usually achieved using contact sulphuric acid plants. The pollutants are captured in combinations of waste heat boilers, cyclones, electrostatic precipitators, wet scrubbers, wet electrostatic precipitators, and demisting towers. The presence of elevated levels of mercury in some concentrates causes difficulties as it is absorbed in the acid product, leading to marketing difficulties. Several remedies are available[5]; the mercury vapour may be removed from the particulate-free gas prior to acid manufacture by dry absorbtion in packed towers containing activated carbon or selenium; by wet scrubbing with concentrated sulphuric acid or mercuric chloride solution, or by H_2S injection and collection of a mercuric sulphide precipitate in coke filters. Alternatively, the contaminated acid may be treated to precipitate dissolved mercury as mercuric iodide, or some other compound, and filtered out. Combinations of pre-removal and acid cleaning may be used. Emissions are greater from multi-hearth roasters when sulphur recovery is not used. Some operations produce dilute SO_2 gases (2%) and these are discharged following particulate recovery. The gases contain fine particulates and fumes which would be captured in the acid plant wet gas cleaning train.

Acid mist is generated during electrolytic refining and the tankhouses have to be ventilated.

Pyrometallurgical refining includes roasting and additional high temperature operations, i.e., sintering, retorting, and possibly distillation refining. Sintering is used to eliminate volatile components which would contaminate the zinc product. Large amounts of lead, cadmium, arsenic and suspended particulates are evolved and high-efficiency particulate control devices, usually fabric filters, have to be used.

Emissions from vertical and electrothermic retorts can be readily controlled using conventional equipment.

Emissions from horizontal retorts are a serious problem. A large number of retorts are required to achieve a reasonable capacity and each retort is a potential emitter. The trend has been to phase out the retorts becuase of the inability to install satisfactory pollution controls.

Fugitive emissions can occur from the processes. These may be minimized through the use of well designed facilities and good operating, maintenance and housekeeping practices. Ventilation systems are installed. Fugitive emissions are more serious for pyrometallurgical refining because of the large number of high-temperature processes involved. Fugitive emissions are very serious from horizontal retorts.

ENVIRONMENTAL LEGISLATION

Various regulatory agencies have issued standards limiting the emission of particulates and certain elements from lead and zinc production, as summarized in Table 4. Other authorities, e.g., some states in the U.S.A., limit the amount of particulates that may be emitted. The limits are normally based on the production rate raised to a fractional power.

Occupational hygiene and ambient air standards also require emissions to be controlled. The U.S. Occupational Health and Safety Administration limit the permissible exposure limits to lead and arsenic in the working environment to 50 ug/m^3 and 10 ug/m^3 respectively over a time weighted eight-hour day[6,7]. This requires particulate and fume emissions to be minimized and/or ventilation provided for operations where lead and arsenic are involved.

The U.S. EPA requires lead emissions to be controlled to meet a limit of 1.5 $\mu g/nm^3$ at the plant boundary[8]. This may require many sources in the plant to be controlled to meet the ground level standard.

Indirect control over particulate emissions is effected through limits placed on SO_2 emissions. Sulphur recovery processes have to be installed to comply with the limits and these require the particulates to be removed for efficient operation.

OUTLOOK

In order to meet the standards summarized in Table 4, substantial changes can be expected, in the next decade, in the processing of lead and zinc concentrates. For instance, weak SO_2 gas-cleaning systems are expensive and the emphasis in the future will be on producing higher-strength gases, which can be treated in, for example, a sulphuric acid plant, in order to meet SO_2 limits such as those set by EPA.

This will necessitate the use of continuous processes and, in the case of lead will result in the increased use of commercial oxygen. The application of the TBRC by Boliden, and the development of the Kivcet process by the U.S.S.R. are examples of this trend in the lead industry.

The Finnish company, Outokumpu Oy, has for a number of years used a flash smelting process for treating copper and nickel sulphide concentrates. At their research station in Pori, Finland, this company has tested, on a pilot plant basis, the application of their flash smelting process to lead sulphide concentrates. It has been found that lead concentrates can be successfully flash smelted but the dust circulation can be high because of the high volatility of lead compounds under these smelting conditions. Nevertheless, a renewed interest in flash and similar smelting processes, applied to lead concentrates, can be expected.

Cominco has done pilot and commercial scale tests on an oxygen flash smelting process producing a soft bullion and a PbO slag (Canadian Patent No. 934,968, issued Oct. 9, 1973).

The Quenau-Schumann-Lurgi (QSL) process is presently being piloted in Germany and could be considered an emerging process. Pelletized concentrates are charged to a reactor. Oxygen and fuel, as required, are blown in through tuyeres mounted in the bottom of the reactor. A relatively high strength off gas is produced which is cooled in a waste heat boiler prior to feeding a sulphur fixation plant. Slag and lead bullion are removed on a continuous basis. The process is still in the early development stages and will require a number of years before it can be considered for commercial operation.

Increasing concern in the industry about the emission of lead to the working environment and ambient air have recently created an interest in hydrometallurgical processes. A number

TABLE 4 SUMMARY OF PARTICULATE EMISSION STANDARDS BASED ON CONTAINMENT AT SOURCE

Agency	Limit	Comments
US - EPA[9]	50 mg/sm^3	Lead blast fce, reverb dross fce, sinter mach. discharge gas
US - EPA[9]	<650 ppm SO_2	Lead sinter machine off gas, (particulate control implied)
US - EPA[9]	50 mg/sm^2	Zinc sinter machines
US - EPA[9]	<650 ppm SO_2	Zinc roaster off gas, (particulate control implied)
Japan[10]	30 mg/nm^3	Lead & Zinc Smelting furnaces (20 mg/nm^3 in more polluted areas)
Germany[11]	20 mg/nm^3	For primary lead plants
United Kingdom[11]	20 mg/nm^3	For large lead works
British Columbia[12]	229 mg/sm^3	Objectives for new facilities
British Columbia[12]	286 mg/sm^3	Objective for existing facilities
British Columbia[12]	343 mg/sm^3	Short-term objective for existing facilities

have been proposed and tested on a limited basis, but none have reached the stage of development of the TBRC and Kivcet processes. The control of intermediate compounds and plant effluents are a few of the problems which must be overcome before the hydro approach can be considered a viable alternative to pyro processes.

The zinc industry has seen more process innovation in recent years than the lead industry, owing in part to the higher value that zinc has commanded historically in the marketplace. This has permitted producers to carry out process development and plant improvements. The electrolytic zinc process, with its various final residue alternatives, removes sulphur as a high strength roaster gas which permits the sulphur to be fixed as sulphuric acid. Hydrometallurgical techniques are then applied to extract the values and precipitate impurities. This approach minimizes particulate emissions, particularly in the form of fume, improves the working environment and provides the type of operation that is amenable to automation.

Developments, therefore, in the zinc industry in the near future will be less spectacular than can be expected in the lead industry. They will be directed towards refinements to the existing processes. The general trend will be away from pyrometallurgical refining processes, except for so-called dirty concentrates, or specialized zinc oxide products. It is reasonable to assume, therefore, that the ISP will continue to have a role in the zinc industry, but other pyrometallurgical processes will be phased out as labour, fuel, and emission constraints increase.

ACKNOWLEDGEMENTS

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CONTROL OF PARTICULATE EMISSIONS FROM THE IMPERIAL SMELTING ZINC-LEAD PROCESS

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ABSTRACT

A brief description is given of the Imperial Smelting Process for producing zinc and lead in a blast furnace. Typical limits on particulate emissions are given for some of the 11 countries where the process is used. The gas cleaning problems met with in various parts of the process are quantified and the methods used to meet the required discharge limits are described. These methods comprise: the use of an electrostatic precipitator or a venturi scrubber to clean acidic sinter plant process gas; the use of a disintegrator wet washing system to clean blast furnace process gas; the use of venturi scrubbers and bag filters to clean sinter and blast furnace plant ventilation gases.

Possible trends in gas cleaning technology applied to the process are discussed.

INTRODUCTION

The Imperial Smelting process for producing zinc and lead in a blast furnace which was developed at Avonmouth, UK, is now in operation at 12 sites in 11 countries. During 1977 a total of 613,000 tonnes of zinc and 282,000 tonnes of lead was produced by the process.

As shown in Fig.1 the first part of the process consists of the updraught sintering of zinc and lead concentrates, returns sinter and fluxes. The concentrates are mainly sulphides and sintering removes the majority of the sulphur as sulphur dioxide and this is converted into sulphuric acid. The product sinter has a typical analysis of 42 - 44% Zn, 18 - 20% Pb, 0.5 - 1.0% Cu and 0.7 - 1% S and is delivered to the furnace bunkers crushed and screened to be 25 - 100 mm in size.

1 CONCENTRATE STORAGE BINS
2 UPDRAUGHT SINTER MACHINE
3 SINTER BIN
4 COKE PREHEATER
5 LEAD-ZINC BLAST FURNACE
6 LEAD SPLASH CONDENSER
7 COOLING LAUNDER
8 ZINC-LEAD SEPARATION
9 GAS WASHING SYSTEM
10 PRIME WESTERN GRADE ZINC
11 LEAD BULLION
12 DECOPPERIZED LEAD BULLION
13 SLAG
14 COWPER STOVES

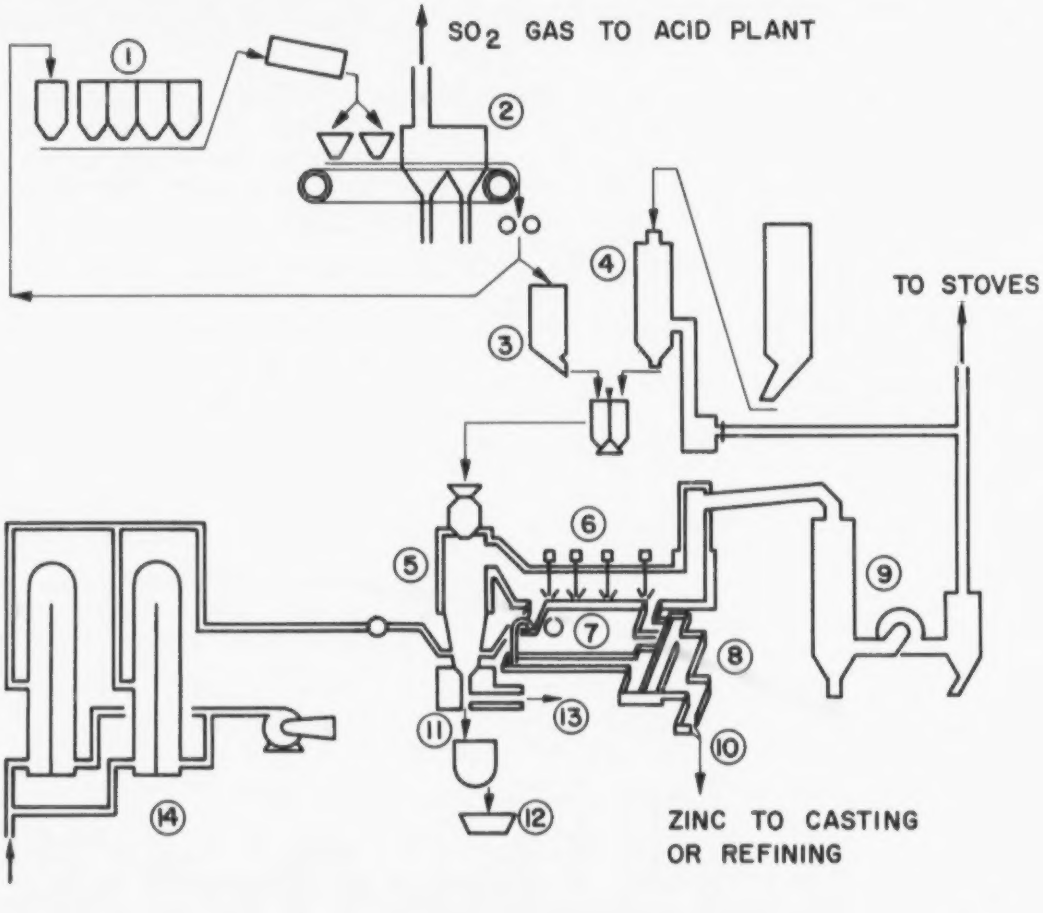
SO_2 GAS TO ACID PLANT
①
②
④
③
TO STOVES
⑥
⑤
⑨
⑦
⑧
⑬
⑪
⑩
⑭
⑫
ZINC TO CASTING OR REFINING

Figure 1. FLOWSHEET OF THE IMPERIAL SMELTING PROCESS.

The Imperial Smelting Furnace (ISF) is charged with coke preheated to 800°C and hot sinter. Inside the furnace the coke is burnt at tuyere level with air preheated in Cowper stoves to 900°C and the product CO reduces the zinc and lead oxides in the sinter. Zinc is produced as a vapour at the furnace temperatures and travels with the furnace gases into a condenser where it is absorbed in a spray of liquid lead. Liquid zinc is then separated from the lead in an external cooling circuit and cast as Prime Western grade zinc or thermally refined to give Special High Grade zinc.

Metallic lead produced in the furnace is tapped periodically from its bottom together with the slag formed from the gangue constituents of the charge. Separation of the lead bullion from the slag is carried out in a forehearth after which the slag is granulated. The lead bullion is either cast into blocks or may be pyrometallurgically or electrolytically refined.

The particulate control systems installed on the process are concerned with cleaning the sinter plant and ISF process gases and also the ventilation gases which are extracted to maintain a suitable working atmosphere. The target emissions to be met vary in the different countries where the process is operated. Some examples are given in Table 1.

Table 1. TYPICAL EMISSION LIMITS FOR PARTICULATES

Plant	Pb mg/Nm^3	Cd mg/Nm^3
Commonwealth Smelting Limited* (CSL) Avonmouth, UK	12	-
Hachinohe Smelting Limited Hachinohe, Japan	10	0.8
Sulphide Corporation Pty.Ltd. Cockle Creek, Australia	20**	-

* CSL is also subject to mass emission limits of 5.4 kg/h for Pb and 0.7 kg/h for Cd

** Total of heavy metals (Pb, As, Sb, Cd, Hg)

Since most of the dust and fumes dealt with contain at least 25% lead, most of these limits are at least as stringent as the EPA limit of 50 mg/Nm^3 for total dust from primary non-ferrous smelters.

SINTER PLANT PROCESS GAS CLEANING

A typical zinc/lead sinter plant produces about 60,000 Nm^3/hr (dry) of gas at a temperature of 300-350°C and containing about 6% SO_2. The fume loading is usually 30-50 g/Nm^3 (i.e. up to 70 tpd) with a typical size range and analysis as given in Table 2.

Table 2. SINTER PROCESS GAS FUME SIZE AND CHEMICAL ANALYSIS

Fume size range	<0.2 μ	0.2-1 μ	1-2 μ	2-5 μ	>5 μ
%	2.3	78.8	8.8	2.9	7.3
Fume analysis	Pb	Cd	Zn	Total S	
%	60-62	3.5-4.5	1-2	10-11	

This gas has to be cleaned and dried before conversion of its SO_2 into SO_3 and the subsequent production of sulphuric acid. The first zinc/lead sinter plants cleaned the gas with venturi scrubbers followed by a cooling tower and electrostatic mist precipitators. Subsequently the hot electrostatic precipitator has become preferred to venturi scrubbers.

Venturi Scrubbers

Venturi scrubbers were successfully used on the prototype updraught sintering machines at Avonmouth and subsequently were applied to large scale machines at 4 sites. At a pressure drop of 600 mm, a gas cleanliness of less than 1 g/Nm^3 is obtainable and this was originally considered adequate in view of the fact that the gas was subsequently cooled and cleaned in an irrigated packed tower and mist precipitators. It has been found in practice that this quantity of particulates has adversely affected the operations of the remainder of the gas cleaning and cooling circuit. A relatively large bleed off from the cooling tower and more frequent washdown of the mist precipitators have been necessary. Even so, some problems with system blockages have occurred. To obtain more efficient gas cleaning a higher pressure drop venturi scrubber would be required. This would not only result in increased power costs but could also have an adverse affect on the life of the fan. The medium head fans used at present only have a life of about 2 years even when constructed from 25% Cr 5% Ni Mo W steel.

Electrostatic Precipitators

Typical resistivity curves for fume from zinc/lead sintering are shown in Fig.2. These curves indicate that electrostatic precipitation should be carried out at a low temperature of say 100°C with saturated gas or at temperatures in excess of 200°C. In the early 1960's the experience which was available on electrostatic precipitation of fume from lead sinter plants had been gained only at low temperatures. This system, which required the installation of a spray conditioning tower ahead of the precipitator was accordingly adopted on several zinc/lead sinter plants. The precipitator had to be lined with acid resistant bricks and further protected with carbon bricks if any fluorine risk existed. Even so the precipitator had a very limited life.

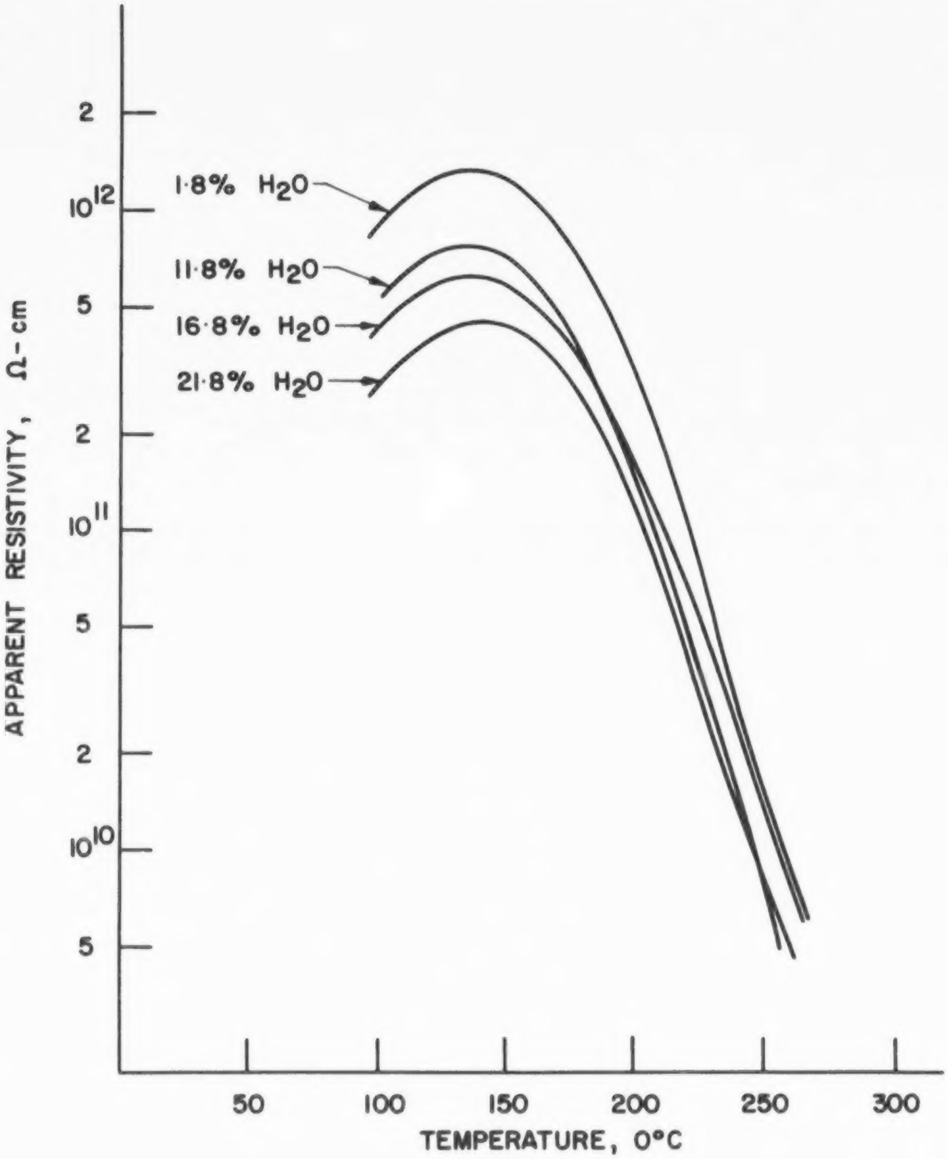

Figure 2. ELECTRICAL RESISTIVITY OF SINTER PROCESS GAS FUME.

As more experience was gained in the operation of large scale updraught zinc/lead sinter machines, it became apparent that sinter gas temperatures of over 300°C could be consistently attained. It was therefore proposed by Lurgi that a hot gas electrostatic precipitator should be used which would considerably reduce the capital costs since an unlined steel cased structure could be employed. Lurgi installed a pilot precipitator on a sinter plant at Avonmouth in 1964 and satisfactorily demonstrated the suitability of a hot gas precipitator for dealing with the fume. The first full scale precipitator was subsequently commissioned in 1966 following which it has become accepted as the preferred method of cleaning sinter plant process gas. Whilst the precipitator has a high capital cost compared with a venturi scrubber,this is outweighed by the following operating advantages:

(1) Low power consumption

(2) High efficiency; this is typically 99.9% giving an outlet dust loading of 20-50 mg/Nm^3.

(3) Few operating problems; such problems as have occurred have mainly been concerned with inadequate electrode rapping and bridging above dust screw conveyors. The latter problem does not appear to arise when drag link conveyors are used.

(4) The overall gas cleaning/acid plant water system is considerably simplified when an electrostatic precipitator is used.

SINTER PLANT VENTILATION GASES

The total ventilation volume on a typical zinc/lead sinter plant is in excess of 150,000 Nm^3/hr. This volume is devoted mainly to ventilating the handling and cooling of product and returns sinter. As shown in Fig.3 the steps involved in this handling system are as follows:

(1) Sinter cake is discharged from the tip end of the machine and crushed by a pronged breaker to <250 mm.

(2) The sinter is fed to a spiked roll crusher which reduces its top size to <100 mm.

(3) The sinter is screened at 25 mm and part of the oversize (30-35% of the total machine output) is fed hot to the ISF charge bunkers.

(4) The remainder of the oversize is crushed in a corrugated roll crusher to <25 mm and is then joined by the screen undersize and the fines recycled from the ISF plant. This material is then crushed to <6 mm by a smooth roll crusher.

(5) The crushed sinter is cooled in a rotary drum by recycled slurries and water and is then transported to the sinter returns bunkers in the feed preparation section*.

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* Note that on some plants the sinter is cooled in the rotary drum before crushing in the smooth roll crusher.

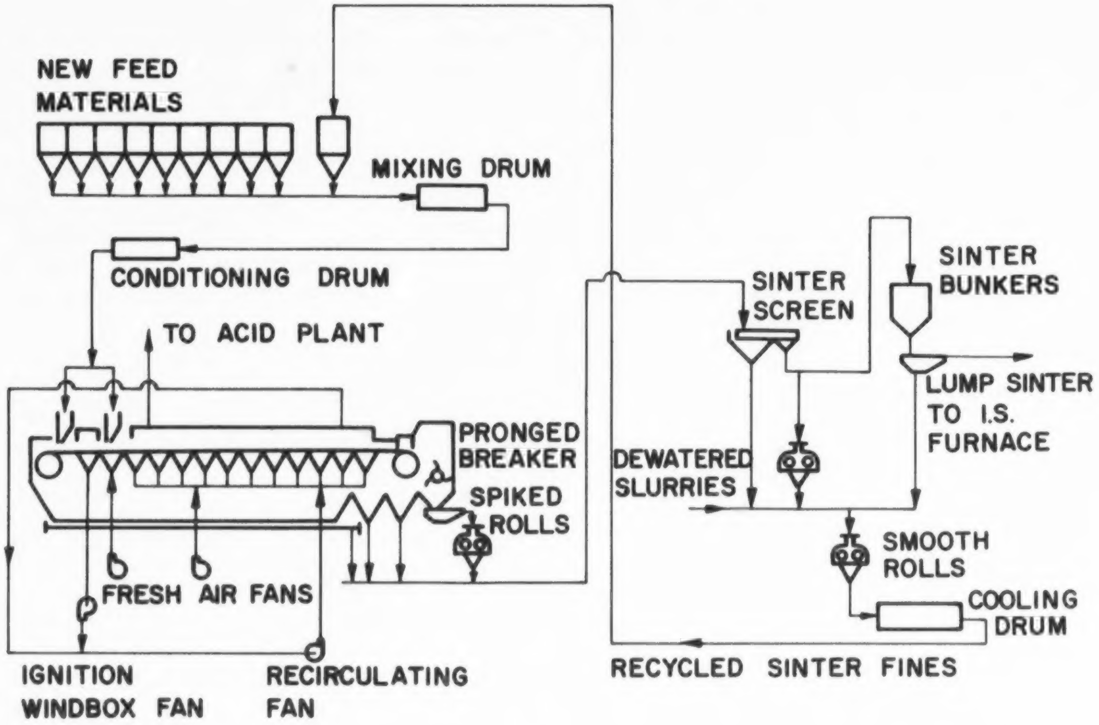
NEW FEED MATERIALS
MIXING DRUM
CONDITIONING DRUM
TO ACID PLANT
PRONGED BREAKER
SPIKED ROLLS
FRESH AIR FANS
IGNITION WINDBOX FAN
RECIRCULATING FAN
SINTER SCREEN
SINTER BUNKERS
LUMP SINTER TO I.S. FURNACE
DEWATERED SLURRIES
SMOOTH ROLLS
COOLING DRUM
RECYCLED SINTER FINES

Figure 3. SIMPLIFIED SINTER PLANT FLOWSHEET.

Tip End Ventilation Gas

This gas, the volume of which is normally about 50,000 Nm^3/hr, is characterized by containing SO_2. The exact quantity of this and the proportion of fine particulates in the gas depend greatly on the extent to which process gas is sucked into the tip end enclosure. By using heat resisting baffles between the machine hood and the enclosure and avoiding pressurizing the machine hood and by ensuring the completion of sintering before the end of the machine, the quantity of fine particulates can be minimized and the SO_2 kept to less than 0.1%.

The following methods have been employed to deal with this gas:

(1) Recirculation to the sinter machine; this requires either that the fan used is specially protected against erosion or that the bulk of the particulates are removed. Recirculation suffers from the disadvantage that the fresh air volume required by the sinter machine is usually less than that required to ventilate the tip end.

(2) Cleaning with a venturi scrubber; this method removes particulates whilst obtaining some SO_2 removal at the same time. However, it has been shown that whilst an outlet particulates loading of less than 50 mg/Nm^3 can be achieved with a venturi pressure drop of about 900 mm w.g., removal of significant quantities of SO_2 is difficult. Unless scrubbing water is used on a once through basis, which is not a practical proposition, then SO_2 can only be removed efficiently by the use of lime slurry dosed recirculated water. This leads to considerable problems with blockages of recycle liquor lines caused by the combined effects of precipitated calcium sulphate/sulphite and the solids scrubbed from the gas.

(3) Removal of particulates by bag filter followed by SO_2 removal by venturi scrubbers; this method has been successfully practised by Hachinohe Smelting in Japan. Data for their bag filter system which handles 54,000 Nm^3/hr is given in Table 3.

Table 3. BAG FILTER FOR TIP END GAS AT HACHINOHE

	Temperature °C	Mean Dust Loading mg/Nm^3
Gas Cooler Inlet	140-170	7 000
Outlet	85-100	1 000
Bag Filter Inlet	81-95	1 000
Outlet	76-85	0.2
Filter Type	Mechanically shaken and reverse air blown	
Bag Material	Acrylic needlefelt	
Filtration Velocity		
gross	0.895 m/min	
net	0.805 m/min	

The venturi scrubbers which follow the bag filter have a pressure drop of 810 mm w.g. and reduce the SO_2 content from 1500 to 200 ppm.

Ventilation Gases from Sinter Handling Systems

These gases generally contain a high particulates loading of over 35g/Nm^3 but the particulates are relatively large as indicated in Table 4.

Table 4. TYPICAL SINTER VENTILATION DUST SIZE

Fume Size Range	<2.5 μ	2.5-5 μ	5-10 μ	10-20 μ	20-30 μ	30-43 μ	>43 μ
%	7.7	6.1	9.7	13.7	13.6	23.1	26.1

Three methods of cleaning this type of dust have been employed:

(1) Impingement scrubbers; scrubbers of the Doyle type were favoured initially for the local ventilation and gas cleaning of individual dust emission points. However, apart from the problems of maintaining a large number of widely separated scrubbers, it was found that these scrubbers were not really suitable for handling high dust loadings. Whilst relatively high scrubbing efficiency could be obtained, daily cleaning of the scrubber was found to be necessary.

(2) Venturi scrubbers; relatively low pressure drop venturi scrubbers have been found to be very efficient in cleaning gases laden with large quantities of coarse dust and are preferred to impingement scrubbers. One example from CSL's plant at Avonmouth is the use of a saturator tower and a 350 mm pressure drop venturi scrubber to clean 35 000 Nm^3/hr of ventilation gases from the product sinter screen and the corrugated roll crusher. The results obtained are given in Table 5.

Table 5. PERFORMANCE OF VENTURI SCRUBBER AT AVONMOUTH

	Total Dust mg/Nm^3	Pb mg/Nm^3	Zn mg/Nm^3	Cd mg/Nm^3
Inlet gas	63 200	16 190	22 570	430
Outlet gas		2.1	1.1	0.16

Venturi scrubbers are also used to clean the gases from the cooling drum. These gases are relatively saturated and the dust burden can contain a substantial proportion of small particles. A 900 mm pressure drop scrubber has been used successfully on this application.

(3) Bag filters; although low head venturi scrubbers are still an economic gas cleaning method, bag filters are used at some sites and are becoming preferred for dry gases since the only liquor circuit possibly involved is that concerned with slurried filter dusts. The majority of bag filters installed have been of the mechanically shaken reverse air blown type. Rozovsky[1] has suggested that pulse jet filters are unsuitable for this type of application because of the high dust loading. However, it is considered to be quite feasible to use such a filter in conjunction with a cyclone.

ISF PROCESS GAS

An ISF with a shaft area of 17.2 m^2 produces about 45,000 Nm^3/h of gas which leaves the lead splash condenser at a temperature of 440-450°C. The gas usually contains 18-23% CO, 9-14% CO_2, 1% H_2 with the balance being nitrogen. The particulates loading is from 20-30 g/Nm^3 with a typical size range and analysis as given in Table 6:

Table 6. ISF GAS PARTICULATES

Size range	< 3 μ	3-5 μ	5-10 μ	10-20 μ	> 20 μ
%	24.3	14.4	20.4	18.4	22.5
Analysis	Pb	Zn	Cd	S	SiO_2
%	30-45	30-45	0.5-2.0	1-3	1.5-2.5

The basic gas cleaning system adopted on all operating furnaces is shown in Fig.4. It consists of a gas washing/cooling tower followed by a disintegrator and a cyclonic moisture separator. A wet gas cleaning system was selected originally because of the explosion hazard with the gas and because of the accreting properties of the particulates, which contain molten lead.

An unpacked gas washing tower is used and at most sites has a central top entry. Two rings of sprays are mounted in the roof of the tower around the gas entry and these are supplied with about 200 m^3/h of recirculated water. The use of this relatively inefficient co-current tower has been dictated by the need to minimize the distance between the condenser offtake and the tower inlet and to avoid any wet/dry transition zone. Tangential entry towers are used by two operators and probably result in better gas cleaning but suffer from accretion build up in the entry duct.

The gas emerging from the bottom of the tower is normally at 40-50°C and contains 5-10 g/Nm^3 of particulates. The gas is fed via irrigated ducts into the inlet legs of the disintegrator. All operators have used disintegrators made by Theisen G.m.b.H of West Germany except the two Japanese plants which have employed disintegrators made by a former Theisen licensee, Ebara Manufacturing Co. of Japan.

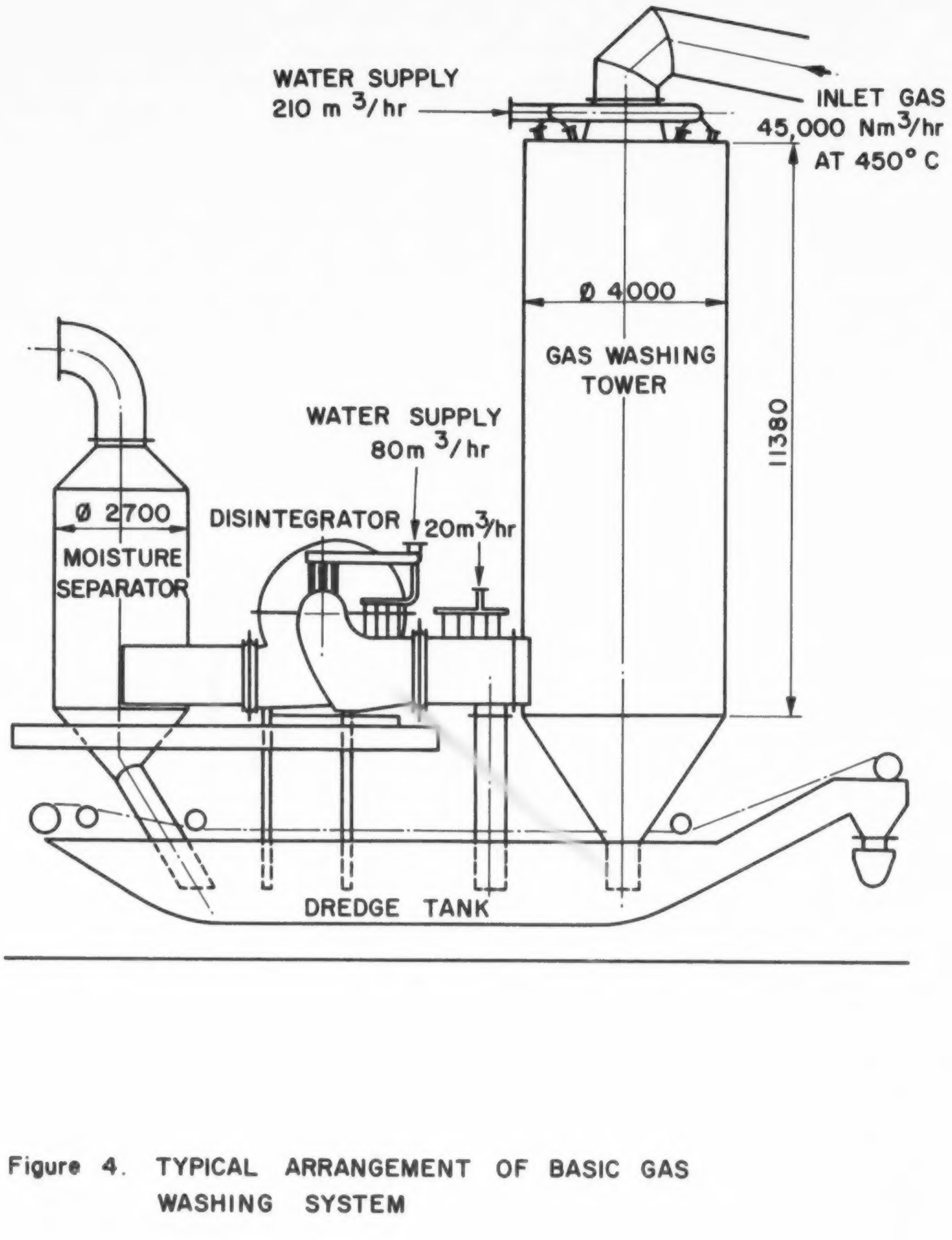
WATER SUPPLY
210 m 3/hr
INLET GAS
45,000 Nm3/hr
AT 450° C
Ø 4000
GAS WASHING
TOWER
11380
WATER SUPPLY
80m 3/hr
20m3/hr
DISINTEGRATOR
Ø 2700
MOISTURE
SEPARATOR
DREDGE TANK

Figure 4. TYPICAL ARRANGEMENT OF BASIC GAS WASHING SYSTEM

A disintegrator is virtually a complicated irrigated fan as can be seen from Fig. 5 which shows the rotor. Water and gas are admitted to the centre of the disintegrator and then pass radially outwards through a series of rings of fixed and rotating bars. The rings of rotating bars are mounted on a rotor which usually revolves at 740 rpm although one plant uses a 960 rpm motor. The high relative velocity between the rotating and static bars smashes the water into very small droplets.

High relative velocities also exist between these water droplets and the particulates and result in good gas cleaning by inertial impaction. The particulates loading of the gas after the cyclonic separator is typically 20-30 mg/Nm^3. At three ISF plants the gas is given further cleaning by a second disintegrator and at another plant a moving ball scrubber is used. Secondary scrubbing results in a particulates loading of 5-15 mg/Nm^3.

The dirty water from the gas washing tower, the disintegrator and the separator is pumped to a thickener and about 75% of it is recycled to the gas washing tower. The underflow from the thickener is filtered and the cake recycled on the sinter plant.

The primary disintegrator has a power consumption of 7-8 kWh/1000 m^3 of gas and a water requirement of 1.2 m^3/1000 m^3. This relatively high power consumption has led to investigations into alternative gas cleaning methods. However, since a wet system is considered necessary then the only alternative would appear to be a venturi scrubber. Some pilot plant work with a venturi scrubber was carried out at the Berzelius ISF in West Germany and gave the results shown in Table 7:

Table 7. VENTURI PILOT SCRUBBER RESULTS

Venturi Pressure Drop mm w.g.	Inlet Dust Loading mg/Nm^3	Outlet Dust Loading mg/Nm^3
800 - 1000	2000 - 3000	80 - 100
1100 - 1200	2000 - 3000	30 - 40
1500 - 1700	2000 - 3000	20 - 25
1600	2000 - 3000	23
1600	5000 - 6000	36

From these results it was concluded that to equal the performance of the disintegrator a venturi pressure drop of over 2000 mm w.g. would be necessary. Even when using a relatively efficient fan, the power required to produce such a pressure drop would have been similar to that required by the disintegrator and there was therefore no justification for changing from its use.

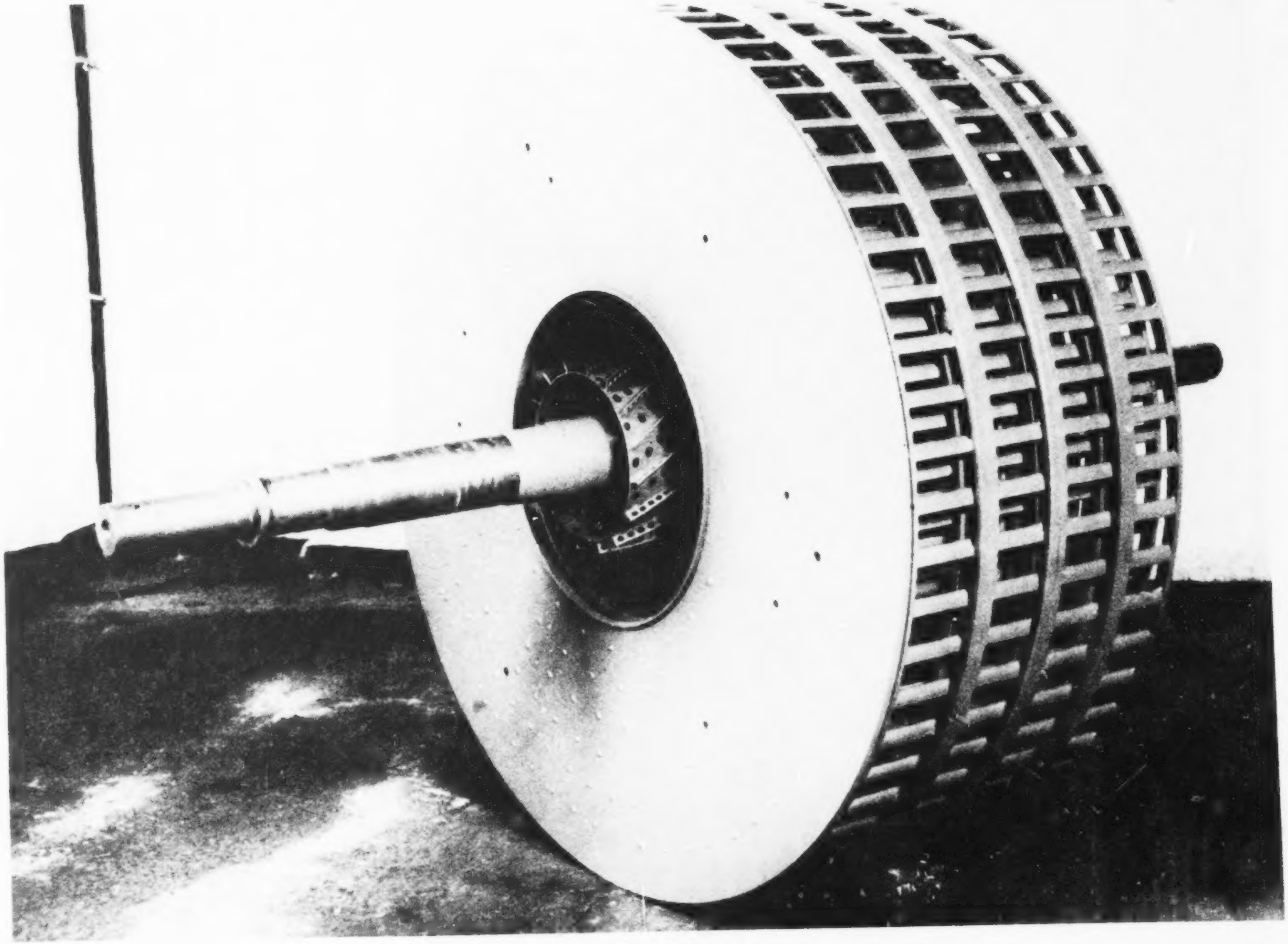

Figure 5. ROTOR OF A THEISEN DISINTEGRATOR.

ISF PLANT VENTILATION GASES

On a typical ISF plant a total volume of about 300,000 Nm^3/h of ventilation gases is extracted. Part of this volume is concerned with ventilating the processes of feeding and screening sinter and coke in the charge preparation section. These ventilation systems at Avonmouth have been described in detail by Senn[2]. The particulates dealt with by the gas cleaning equipment are basically similar to those dealt with in the sinter plant sinter handling systems and the same remarks about gas cleaning systems apply.

The remainder of the ISF ventilation systems are largely concerned with molten metal and slag handling systems.

Zinc Separation Systems

In this system, lead at 550°C containing about 2.3% zinc is pumped from the condenser into a refractory lined launder where it is cooled by immersible water-cooled steel coolers. The temperature of the lead is progressively reduced to about 440°C as it travels along the launder with the effect that the solubility of zinc is reduced and molten zinc separates from the lead. At the end of the launder the lead/zinc liquid mixture flows into a flux bath and then into a large bath where the separation process continues in quiescent conditions. Zinc is then recovered by overflowing into a holding bath whilst lead returns to the condenser via an underflow weir.

Since the temperatures in this system are relatively low, metallic fume formation is not a major problem. The fume extraction system is primarily applied to ventilate positions where zinc and lead drosses are removed and also to remove products of combustion from the separation baths. However, the removal of ammonium chloride fume from the flux bath is an important ventilation requirement.

It has been found that a venturi scrubber with a pressure drop of 450 mm w.g. has enabled satisfactory discharges to be maintained in this relatively lightly loaded system. Corrosion of the ductwork after the scrubber caused by ammonium chloride and its dissociation products is a problem. Several ISF plants have installed bag filters instead of scrubbers to clean these gases.

Slag and Lead Bullion System

Slag and lead are tapped from the bottom of the furnace at intervals of 1¼ - 1½ hours and flow for 20 - 30 minutes. The slag, at temperatures of up to 1300°C, flows with the lead into a forehearth. Separation takes place by gravity with the slag overflowing and being granulated by water whilst the lead syphons out of the forehearth via an underflow weir and is then run into a ladle. A typical arrangement of this area is shown in Fig.6.

The cleaning of the ventilation gases from this area has been described in some detail by Lee and Coy[3]. It was shown at Avonmouth that the 600 mm w.g. pressure drop venturi scrubbers originally installed could only obtain a gas cleaning efficiency of 92-95% compared with the level of 99% which was

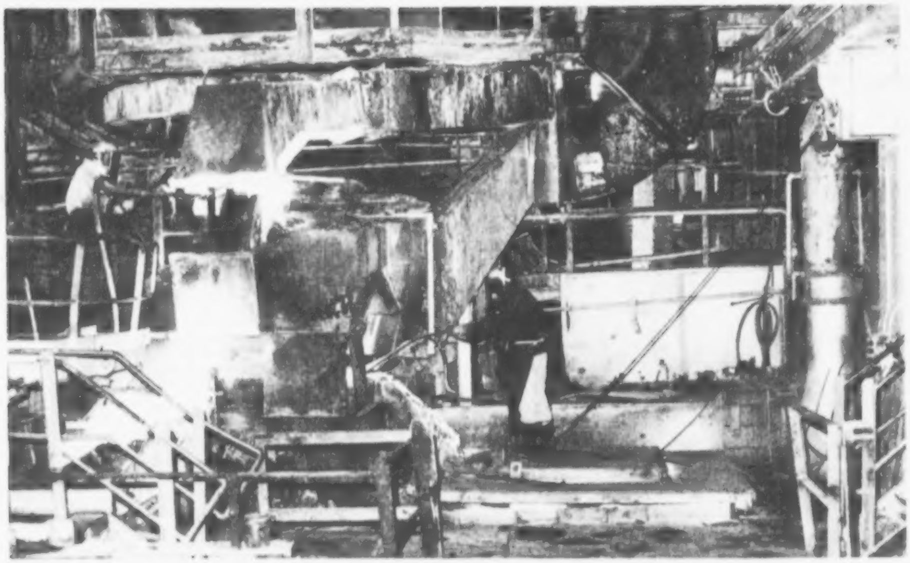

FIGURE 6. SLAGGING FLOOR VENTILATION AT AVONMOUTH

required to meet the discharge regulations. Size analysis indicated that over 14% of the fume was sub-micron in size. It can be seen from Fig.7 that the majority of the individual particles are sub-micron in size but may form agglomerates.

It was concluded that to clean the gas satisfactorily either a venturi scrubber with a pressure drop in the region of 2000 mm w.g. was necessary or that a bag filter should be installed. After some trial work a pulse jet filter was installed as illustrated by the photograph in Fig.8. This unit is equipped with envelope bags arranged in four tiers and made from 645 g/m^2 terylene needlefelt. The filter, which is pulsed on line, has a specific filtration velocity of 1.33 m/min . The fume collected by the filter is slurried and recycled to the sinter machine via an existing dewatering system. Results obtained with the filter are given in Table 8:

Table 8. SLAGGING FLOOR FILTER PERFORMANCE AT AVONMOUTH

Volume handled Nm^3/h	Position sampled	Fume Concentration mg/Nm^3				
		Total	Pb	Zn	Cd	As
41,850	Inlet	8,118	3,555	2,977	5.5	28.9
	Outlet	-	0.40	0.35	0.03	0.13
41,120	Inlet	7,046	1,685	4,322	2.7	17
	Outlet	-	0.23	0.37	0.17	0.008

There has been a general trend by ISF operators towards the use of bag filtration for cleaning slagging floor fumes. However the use of a venturi scrubber is still necessary for dealing with the fairly saturated fume produced when the slag is granulated with water.

FUTURE DEVELOPMENTS IN GAS CLEANING TECHNOLOGY

In common with the other non-ferrous metals extraction processes the Imperial Smelting process is being subjected to increasingly stringent environmental regulations. Since these regulations cover the working atmosphere, gaseous discharges and liquid effluents the following effects can be anticipated:

(1) To meet tighter threshold limit values for various pollutants increased ventilation volumes will be required despite efforts to improve process systems and to use more effective hooding.

(2) Gases will require cleaning to higher levels of efficiency. Removal of sub-micron sized fume particles will become increasingly important.

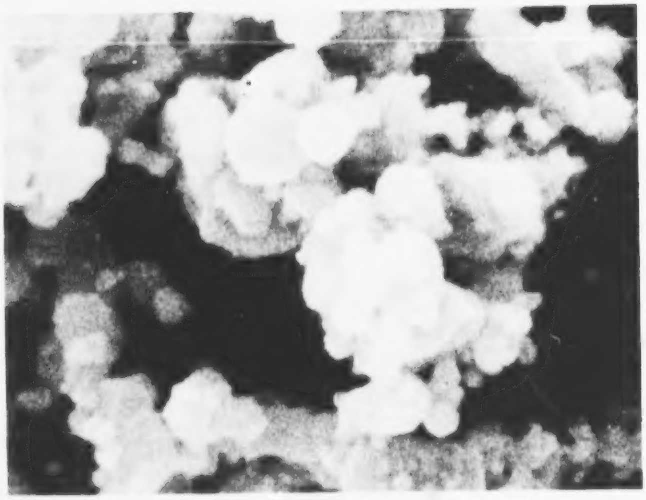

Figure 7. SLAGGING FLOOR FUME (Photo courtesy of Skandiafelt A.B.,Sweden)

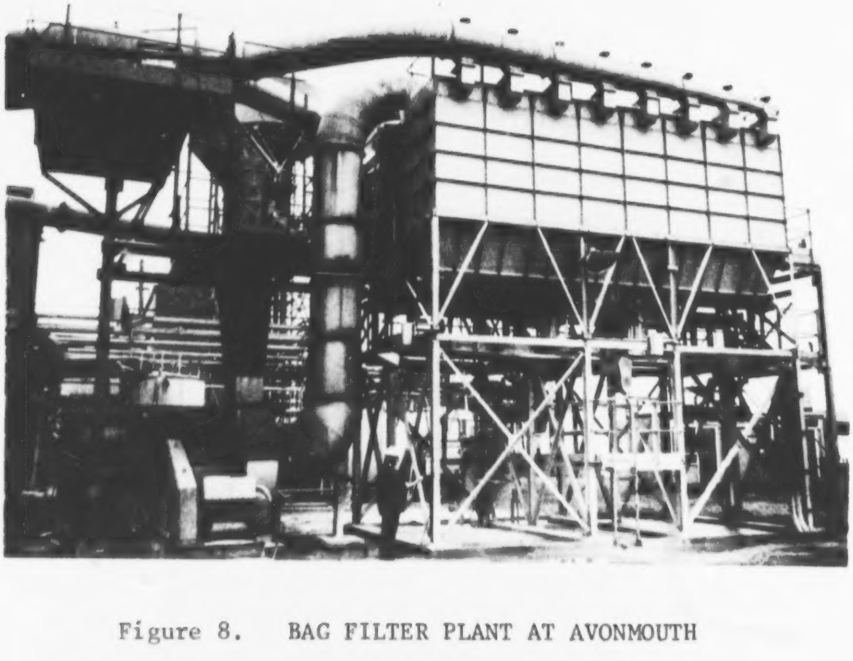

Figure 8. BAG FILTER PLANT AT AVONMOUTH

(3) The trend towards the use of the bag filter for gas cleaning will continue:

  (a) It is the most effective method on sub-micron sized fume.

  (b) It presents no liquid effluent treatment problems.

  (c) Maintenance problems have been reduced by improvements in filter design and bag materials.

  (d) There would appear to be further scope for improvements in filter design and bag materials which would result in reduction in the capital costs of filters.

(4) Venturi scrubbers will continue to be used extensively particularly where:

  (a) Moisture laden gases are to be cleaned.

  (b) Removal of SO_2 is a requirement.

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Theisen G.m.b.H Munchen, West Germany.

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PARTICULATE CONTROL AT THE PORT PIRIE LEAD SMELTER

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Port Pirie, South Australia 5540

ABSTRACT

A lead smelter has operated at Port Pirie since 1889 to treat the rich lead-silver-zinc ore mined at Broken Hill. The present production capacity is in excess of 200,000 tonnes per annum, using a conventional updraught sintering machine, blast furnace and lead refining units. Slag fuming furnaces and an electrolytic zinc plant were installed in 1967 to recover the zinc from the lead blast furnace slag. Over the years, considerable development has taken place in improving the control of particulate emissions from the smelter, and installed collection equipment includes both fabric filters and wet scrubbers B.H.A.S. experience with the various types of collectors is discussed together with current views on equipment suitable for a lead smelter.

INTRODUCTION

The development of the lead and zinc industry at Port Pirie, South Australia, dates from the discovery of the rich lead-silver-zinc mining field at Broken Hill in the state of New South Wales, in 1883. As the nearest sea port to Broken Hill, Port Pirie provided a natural outlet for the products of the field and following the erection of a lead smelting plant in 1889, has continued to treat the major share of the lead concentrates produced. Port Pirie is situated on the eastern shore near the head of Spencer Gulf in South Australia, approximately 230 kilometres from the State Capital, Adelaide and 400 kilometres from the inland city of Broken Hill.

High grade lead sulphide concentrates containing about 75 percent lead provide 85-90 percent of the input materials to the plant while the balance is made up of purchased scrap battery plates and other minor lead bearing materials. The lead concentrates contain approximately 3.6 percent zinc which is collected in the lead blast furnace slag. By 1967 some 6 million tonnes of slag containing around one million tonnes of zinc had accumulated in the slag dump and a treatment plant was then built to recover this zinc.

The annual production capacities of the main sections of the plant are as follows:-

Lead	230 000 tonnes
Silver	280 000 kilograms
Zinc	45 000 tonnes
Sulphuric Acid	90 000 tonnes

Other by-products include gold, cadmium, copper matte and antimonial lead alloys.

Particulate emissions from pyrometallurgical processes are substantial and the combination of small particle size and high carrier gas temperatures make collection difficult. In addition the toxicity of lead bearing materials to man is well known and the ever decreasing emission limits being imposed by environmental authorities makes it mandatory to utilise the most efficient collection equipment available.

SOURCE AND TYPE OF EMISSIONS

The major particulate emissions in the lead smelter arise from the sintering, blast furnacing and refinery processes and are generated in sufficient quantities to be economically recovered. While the gaseous constituents of these emissions vary considerably between processes, the particulates have the common property of being very finely divided with a significant proportion in the fume category. Attempts have been made to conduct sizing analyses on the different fumes but some difficulty has been experienced due to agglomerates breaking up on impact. However, meaningful figures for blast furnace fume and fume from dross kettles etc. have been obtained which showed that the fume was almost entirely sub-micron size confirming data obtained elsewhere that high temperature smelting processes generate mainly sub-micron fume with a lead content in excess of 60 percent.

The sinter plant process gases are divided into two streams, the smaller fraction containing up to 7 percent sulphur dioxide being delivered to the acid plant while the balance containing up to 0.7 percent sulphur dioxide is at present filtered in two baghouses. High pressure water sprays are used to control the temperature at 200°C in the machine hood and the gas temperature at the baghouse inlet is maintained at about 100°C. A water dew point of up to 50°C has been measured and any condensate is acidic and severely corrosive. A dry electrostatic dust precipitator followed by wet electrostatic mist precipitators are used in the high sulphur dioxide stream going to the acid plant. Dust loadings vary up to 16 grams per cubic metre of gas.

Blast furnace gases are induced into an overhead hood and together with other process and hygiene gases from the copper removal stage in the refining operation are filtered in a baghouse. The sulphur dioxide content of blast furnace gases is about 0.25 percent and the carbon monoxide level can reach 1 percent. A lower dew point in the 15 to 20°C range means that less corrosion problems are experienced in this baghouse. However, the fume will at times ignite at about 140°C and problems have been encountered with small smouldering pockets of fume in the collecting launders which have been caused by a hot screw conveyor bearing. Dust loadings in blast furnace gases can be up to 6 grams per cubic metre but there is a drop out along the main smelter flue which is a low velocity regime. Refining process gases are also handled in the blast furnace baghouse, but they consist mainly of oxide fume with little moisture or sulphur dioxide present.

A fume containing up to 50 percent arsenic, 10 percent lead and 20 percent antimony is volatilised in a reduction process at the lead refinery and is collected in a small baghouse. The volume of gas handled is only about 750 cubic metres per minute, but extra precautions are taken to protect the health of operating personnel.

Hygiene gases draughted from belt conveyors, transfer points and crushing rolls in the sinter plant have caused problems due to the wet sticky nature of the particulates. Lead dust is very difficult to keep in suspension if being transported in a wet gas as it tends to settle out in the duct system, causing flow restrictions. A wet scrubber is used to collect the particulates under these conditions and the collection units must be installed close to the emission source.

Particulate collection is also important in our zinc division, and a baghouse is used to recover the zinc oxide fume liberated in the slag fuming furnaces. The baghouse is similar to our lead division process baghouses except that higher gas temperatures up to 230° are permitted by the use of Nomex filter fabric. Dust loading is high at 65 grams per cubic metre.

Emissions from the zinc oxide roasting kilns have caused problems as the high fluorine and chlorine content of the gases precludes the use of a baghouse in this location. The particulate loading is about 25 grams per cubic metre and it is a sticky fume causing rapid build up in the flue system. Two short throat venturi scrubbers in series are used to collect the particulates but emission levels are still unsatisfactory.

A kiln is also used to dry the residual material obtained after leaching out the zinc from the zinc oxide fume and the emission control in this unit is unsatisfactory. The gases are hot, moist and contain a high proportion of sub-micron particulates. A low energy microdyne wet scrubber is installed, but it is inadequate for the duty.

Electric furnaces are used to melt the cathode zinc produced in the electrolytic plant and problems were encountered with the fume emissions resulting from the use of ammonium chloride as a flux. Tests to control emissions with a microdyne wet scrubber were unsuccessful and a reverse pulse small baghouse was finally installed.

Small baghouses are installed to collect the product produced in the zinc dust furnace and the zinc oxide dust recovered in the zinc dross treatment plant but these are standard installation with no particular problems. Existing sources of particulate emissions together with the type of collecter installed are summarised in Table 1.

PARTICULATE CONTROL AT B.H.A.S.

The regulatory authority responsible for controlling gaseous emissions in the state of South Australia is the S.A. Health Commission, and control of particulate emissions is achieved through the Clean Air Regulations, 1972 under the Health Act, 1935-71. These regulations took effect on 1st January, 1973 but included provision to obtain exemptions at the discretion of the authority. B.H.A.S. meets the requirements in most cases but there are still areas of concern where exemptions are being obtained on an annual basis.

Control standards are established at the emission source and standards of concentration of air impurities are expressed as the amount of such impurity in each cubic metre of effluent gas before admixture with air, smoke or other gases. The regulations applying to the smelter operation at Port Pirie specify the following emission concentration limits on a dry gas basis at 0°C and 760 millimetres pressure.

Particulates - not to exceed 110 milligrams in each cubic metre of effluent gas.

Heavy Metals - total mass of copper, lead, arsenic, antimony, cadmium mercury or any of their compounds shall not exceed 20 milligrams in each cubic metre of effluent gas in existing plant or 10 milligrams in each cubic metre of effluent gas for all new plant additions.

General Philosophy

The first filtering device at the Port Pirie smelter was a baghouse installed in 1907 and since this time there has been a progressive increase in the use of bag filters for gas cleaning. We consider fabric filters to be a most effective means of capturing lead fume from pyrometallurgical activities and providing a reasonable bag maintenance programme is maintained there is no problem in meeting the Clean Air Regulation standards. The policy is to use dry filtration equipment on the Works wherever possible.

Our experience with wet scrubbers has not been encouraging but the problems have been caused more by the difficult nature of the dust handled than by the collection unit. In the lead smelter, wet scrubbers have been used mainly to treat wet sticky gases in the sinter plant, and the sticky nature of the collected particulates causes operating problems. The solids have accumulated in the ducts and scrubbers to the point where draughting is reduced and the plant has to be shut down for cleaning. However, up to the present time we have been using low energy scrubbers not suited to the type of dust handled and recently-installed venturi scrubbers are performing better. Emission standards are being met and cleaning problems are reduced significantly. A different problem exists at the zinc oxide roasting kiln of the zinc plant where the installed high energy venturi scrubber is not meeting emission standards. Efforts are being made to improve this performance and a longer throat venturi will be tried in this position.

An electrostatic precipitator is installed to collect the majority of particulates contained in the sinter plant gases used for sulphuric acid production, but the collection efficiency of 93 percent is too low for general use in other plants.

Baghouses

There are many factors to be considered when designing a baghouse for handling large volumes of dirty process gases but the operating principles are relatively simple. Many plants have developed their own design from practical experience acquired over many years and this is certainly the case at Port Pirie. We are at present constructing a centralised baghouse system to handle most of the lead plant process gases, as part of an environmental project costing about $23 million. The project revolves around the construction of a 205 metre tall stack designed to provide sufficient dispersal of the residual sulphur dioxide in effluent gases to meet the Clean Air Regulation limit of a maximum ground level concentration of 20 parts sulphur dioxide per hundred million of air over a period of 3 minutes. A diagram showing the general gas flow arrangement is shown in Figure 1 and the project is expected to be operational in July, 1979.

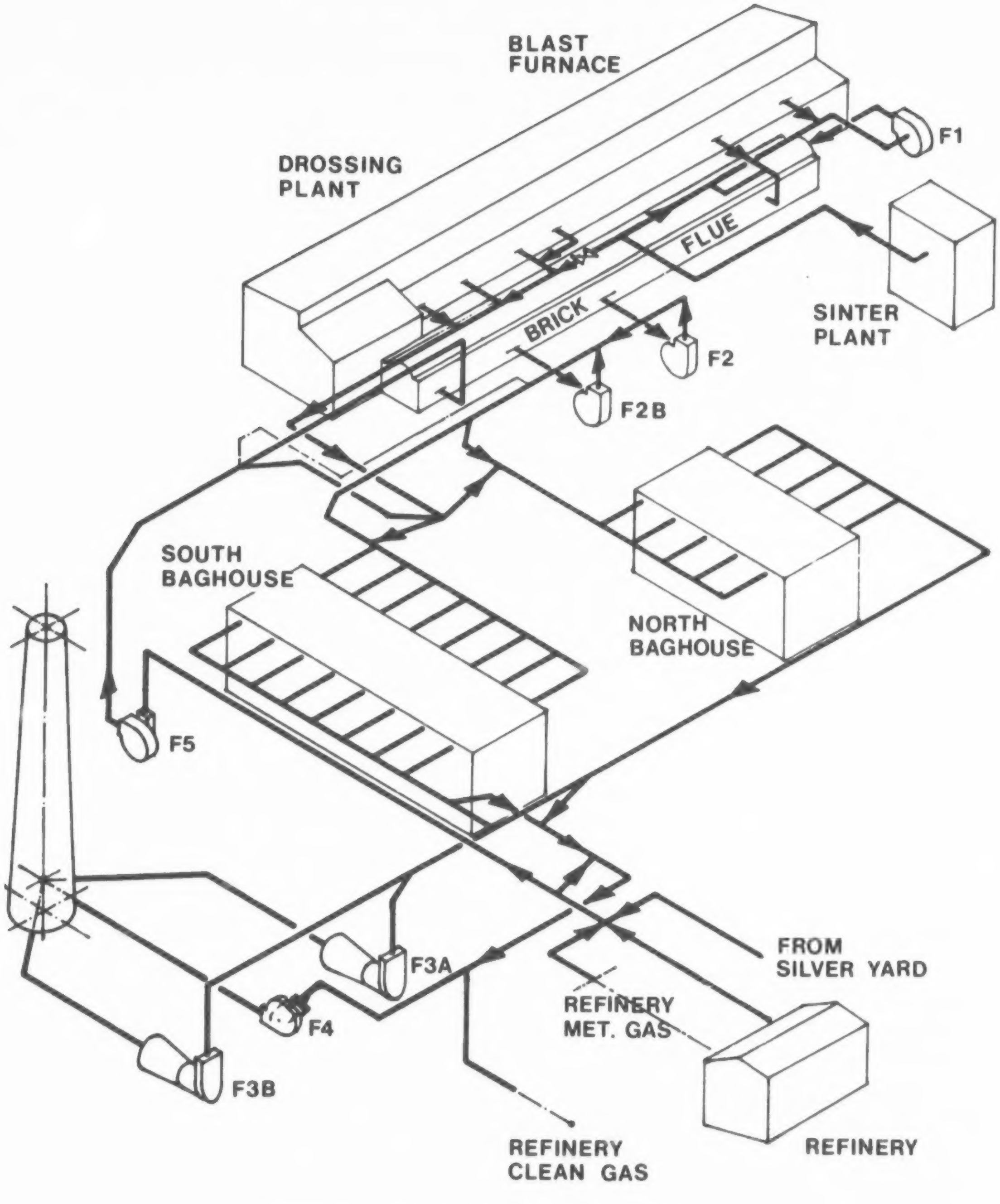
BLAST
FURNACE
DROSSING
PLANT
F1
FLUE
BRICK
SINTER
PLANT
F2
F2B
SOUTH
BAGHOUSE
NORTH
BAGHOUSE
F5
F3A
F4
F3B
FROM
SILVER YARD
REFINERY
MET. GAS
REFINERY
CLEAN GAS
REFINERY

Figure 1 — General Gas Flow Diagram

Table 1. SOURCE OF EMISSIONS AND TYPE OF COLLECTOR INSTALLED

Source of Emission	Vol. Handled nm^3/min.	Type of Collector	Particulate Loading g/m^3	Collect. % Effic.
Blast furnace and refinery process gases	6770	Bag filter (mechanical shaking)	2.7	99.3
Sinter plant gases (lower sulphur dioxide)	4630	Bag filter (mechanical shaking)	4.0	99.5
Sinter Plant gases (high sulphur dioxide)	680	Dry electrostatic precipitator	16.0	93.0
Sinter plant hygiene gases	180	Venturi Scrubber (medium energy)	1.4	99.8
Lead refining process gases (high arsenic)	750	Bag filter (reverse air pulse)	1.8	99.2
Slag fuming process gases	3070	Bag filter (mechanical shaking)	65.0	99.9
Zinc Oxide roasting kiln gases	240	Venturi Scrubber (high energy)	25.0	98.8
Leach residue drying kiln gases	130	Microdyne scrubber (low energy)	4.0	91.0
Zinc melting furnace gases	350	Bag filter (reverse air pulse)	0 - 7.0 (Variable)	Not Calculated
Zinc dust manufacture gases	110	Bag filter (mechanical shaking)	3.3	Not Calculated
Zinc dross treatment gases	450	Bag filter (mechanical shaking)	10.9	Not Calculated

The design parameters applied to the new baghouse under construction are discussed in succeeding paragraphs and they incorporate the factors we feel are important in designing a baghouse system for filtering and recovering particulates from a lead smelter process gas. A sketch of a typical baghouse chamber layout incorporating a mechanical shaking arrangement is shown in Figure 2. We also have several smaller packaged baghouses using mechanical or reverse jet pulse shaking mechanisms and these will be discussed separately.

General Design - A standard compartment (Chamber) size of 17.5 metres x 4.5 metres x 11.5 metres high has been adopted for the lead smelter in which it is possible to hang 530 filter bags, 229 millimetres in diameter by 8.84 metres

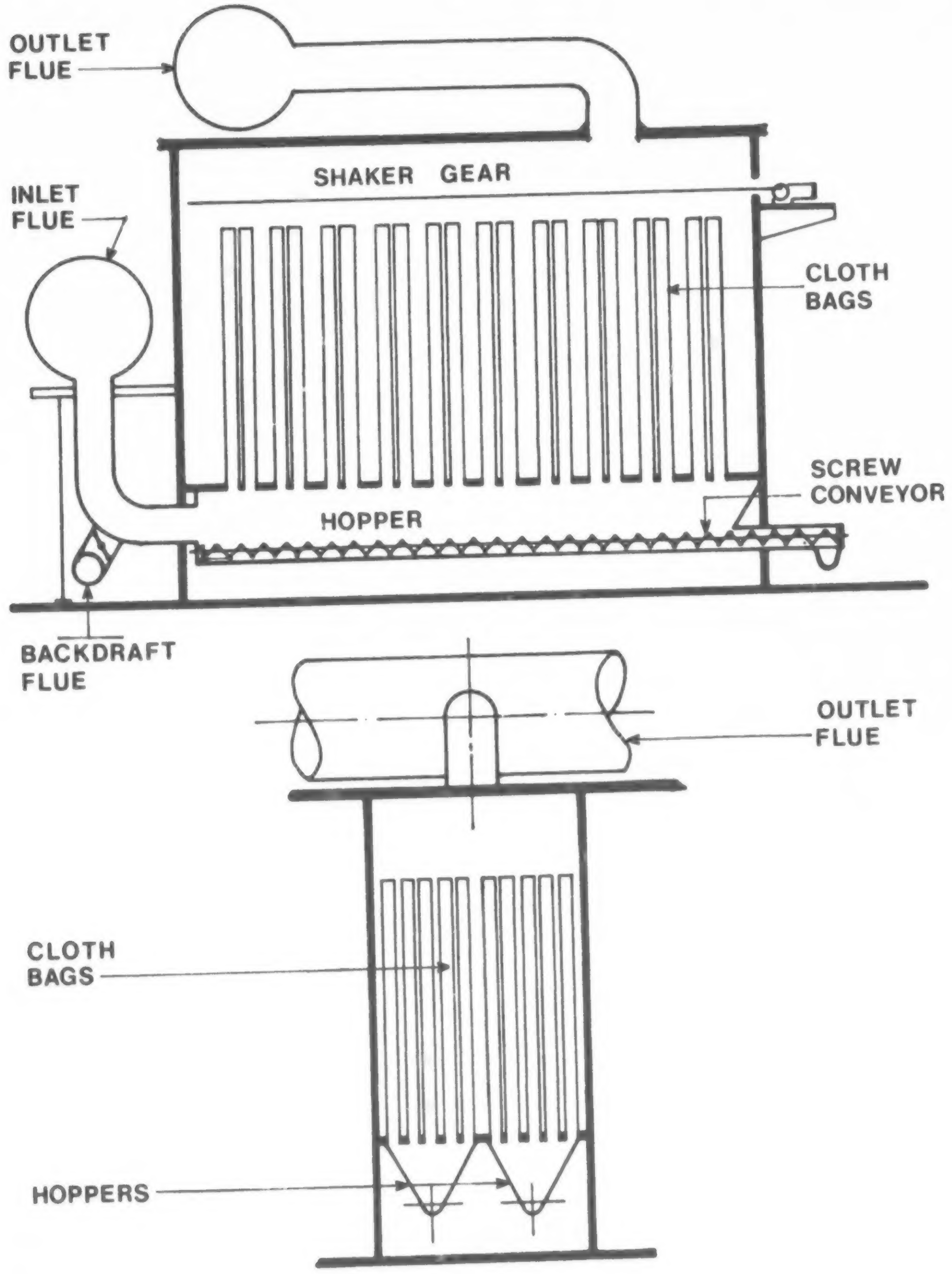
OUTLET FLUE
SHAKER GEAR
INLET FLUE
CLOTH BAGS
SCREW CONVEYOR
HOPPER
BACKDRAFT FLUE
OUTLET FLUE
CLOTH BAGS
HOPPERS

Figure 2 — Typical Baghouse Chamber Layout

long. Each bag has an effective filter area of 6.2 square metres, giving a total filtering area of 3286 square metres per chamber.

There will be thirteen such chambers in the centralised baghouse system in two separate buildings. One building (existing) comprises eight chambers while a new building of five chambers is being built, with provision for extension later if required. The new building will be cast in concrete as opposed to mainly brick construction used in previous baghouses. A mastic material called Stackfas has been thickly sprayed on to the inner walls to reduce the possibility of corrosion by acidic condensate which may form when a chamber is taken off line for checking. Corrosion does occur at times in the present sinter plant baghouses if all openings are not sealed properly and cold outside air is allowed to enter the chamber. The temperature of gases from the sinter plant off-take will be raised to a maximum of 350°C and the moisture content lowered so that the mixed sinter plant - blast furnace gases will enter the baghouses at 110°C - 120°C and this should give less condensation problems.

Close attention has been given to reducing frictional losses and maintaining streamline gas flow wherever possible in designing the new ducting and damper system, as tests have shown that in the past much power has been wasted in overcoming unnecessary resistance from these factors.

The fume dislodged from the filter bags falls into vee shaped hoppers at the bottom of the chambers from where it is wormed out by screw conveyors set in the hopper. It discharges into another collector screw conveyor outside the chamber and is eventually mixed with cadmium plant filtrate in a sump and returned to the sinter plant circuit.

A back draughting offtake is connected into the inlet duct of each chamber on the baghouse side of the inlet damper. Its purpose is to draught away any gas which may leak past a closed inlet damper while operators are working in a chamber. The sulphur dioxide and carbon monoxide content of smelter gas are sufficiently high to warrant this safety precaution.

Balanced Draughting System - A balanced draught will be maintained over the baghouse in the new sytem. This means that two sets of fans will be required, one delivering dirty gas to the baghouse and the other taking the clean gas after filtration. The aim will be to maintain the null pressure point in the bottom hoppers of the chambers where the dirty gas enters.

The decision to install a balanced draught system was not easy and was not unanimous, but it followed a recommendation from the Fume Superintendent after he had studied the practice of other overseas plants and compared it with our own experience. Our present blast furnace baghouse works under balanced draught conditions and apart from the inconvenience of having to clean the dirty side fan blades about once each month, is regarded as satisfactory. On the other hand, the existing sinter plant baghouses work under higher suction (induced draught) conditions and serious corrosion problems have been encountered in the chambers due to entry of cold false air through poorly sealed doors and openings. The high dew point and sulphur dioxide content of sinter plant gases contributes to the problem. False air leakage results in a greater volume of gas to be handled by the fan in an induced draught system and to some extent will counterbalance the extra power requirements for a balanced draught arrangement.

Two fans in parallel will handle the dirty gases from the smelter flue into the baghouses and they will be controlled automatically to maintain a set suction level in the flue. They are centrifugal fans each designed to handle 200 cubic metres per second of air at 352 revolutions per minute shaft speed and driven by a variable speed 500 kilowatt electric motor. The fans on the clean side of the baghouse will each be axial flow type designed to handle 210 cubic metres per second of air at 735 revolutions per minute shaft speed and driven by a variable speed 850 kilowatt electric motor. Suction on the baghouse outlet will control these fans and there will be a feed forward control system link between the fans on the inlet and outlet sides of the baghouses.

An emergency diesel generator will be installed on the axial flow fans to provide minimum draughting in the event of a general power supply failure.

Air to Cloth Ratio - For baghouses using a mechanical bag shaking system, an air to cloth ratio (filter velocity) of 5 millimetres per second has been used at Port Pirie in the past, but testwork has shown that this can be increased to 9 millimetres per second without increasing particulate emission levels beyond the regulation limit. The higher figure has been used in the centralised baghouse system for an expected average pressure drop across the bags of about 80 millimetres water gauge.

A higher air to cloth ratio of 12 millimetres per second has been found satisfactory in small prefabricated baghouses using a reverse pulse of compressed air for cleaning the bags.

Filter Cloth - The original filter fabric used at Port Pirie was a plain weave woollen cloth but over the intervening years tests have been conducted in operating baghouses trying out different types of cloth, different weaves and the effect of a raised surface on one side of the cloth. As a result of these practical tests it was found that polyacrylic cloth performed best in the sintering plant baghouses where the sulphur dioxide and moisture content of the gas is highest. A bag life of approximately 3 years is obtained at present in this location. The specifications of the cloth in use (Dralon, Orlon, Microtain) fall within the following limits.

Weave - 2 x 2 twill
Weight - 187 to 250 grams per square metre
Permeability - 14 to 17 cubic metres per square metre per minute at 25 millimetres water gauge pressure.

Polyester cloth is preferred in the blast furnace baghouse due to its slightly better abrasion resistance and lower cost, and will be used in the centralised baghouses handling the mixed gases. These bags have a life of approximately 7 years in the present baghouse. The terylene cloth used has the following specification -

Weave - Irregular twill
Weight - 224 grams per square metre
Permeability - 16 cubic metres per square metre at 25 millimetres water gauge pressure.

However, since concern has been expressed over the potential harm that fluorocarbons may cause to the ozone layer, care must still be taken to avoid accidental release of Freon 12. On the other hand, because Freon 12 completely breaks down at molten aluminum temperature, typically above 650C (1200F) there is no risk of introducing unreacted Freon 12 into the atmosphere as a result of the fluxing process. This is supported by data (12) that indicate that Freon 12 in quartz starts to decompose at 540C (1000F). Since it is also known (14) that aluminum lowers the stability of Freon 12, it is very unlikely that any Freon 12 can withstand the temperatures encountered in aluminum alloy melts.

The possible existence of C, Cl_2 and N_2 in presence with moisture (H_2O) in the air also raises the concern of forming cyanide CN and/or phosgene $COCl_2$, both highly toxic compounds. However, measurements presented later will show conclusively that this is not the case.

Emission Sampling Procedure

Throughout the experimental development and initial plant implementation of the nitrogen-Freon 12 melt treatment process, continuous sampling was performed for particulate and soluble halide gases. Sampling took place in the stack, close to the roof line of the cast house, with a gas train* and procedures (15) recommended by the Federal Environmental Protection Agency. The sampling was isokinetic in that the average velocity of the sample stream was the same as the average gas velocity within the stack. Depending on the stack cross-section, sampling was conducted at either one central location in the smaller stacks or 4 to 8 locations in the larger ones.

The solid particulates were weighted and fired at 1000C (1830F) for 1 hour. The aluminum was analyzed and reported as Al_2O_3, the remainder consisting of oxides which were not analyzed qualitatively and were reported with the ignition losses. The caustic and water solutions used to trap soluble halide gases were also analyzed for Al^{+3}, Cl^- and F^-. Since negligible quantities of Al^{+3} were found, results were reported on the basis of emission rates of HCl and HF. Carbon content was analyzed when Freon 12 was used. In this case, a Dragon technique was also used to measure phosgene and carbon monoxide levels; cyanide was checked for with a colorimetric technique using a cyanide test outfit.**

Experimental Development of Inert Gas-Freon Melt Treatment

Initial experimental work was carried out on 450 lbs of induction melted pure aluminum (99.85% Al) at a total gas flow rate of 15 l/min (0.53 scfm).

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* RAC Model 2343 Train Staksampler, Research Appliance Co., Allison Park, Pennsylvania.

** Model SCF, LaMotte Chemical Products Company, Chestertown, Maryland

This brought out two major conclusions:

1. Nitrogen-10% Freon 12 and argon-10% Freon 12 gas mixtures were more efficient than either N_2-10% CO-10% Cl_2 or N_2-10% Cl_2 in terms of degassing rate.
2. Nitrogen-10% Freon 12 and argon-10% Freon 12 were more efficient than pure chlorine.
3. A molten salt cover flux was effective in suppressing fume emission and increasing degassing rates.

Subsequent pilot plant tests with a gas fired furnace and 4500 lbs melts, again of pure aluminum (99.85% Al) showed that Freon 12 levels in the 5-10% range, lower than previously thought, degassed as effectively as pure chlorine. The total flow rate was then 90 l/min (3.2 scfm) and no flux cover was used. The emission rates shown in Table 1 and illustrated in Figure 1 indicate a significant reduction in emission rate, for an equivalent degassing efficiency, as a result of using N_2-10% Freon 12.

Table 1 - Emission rates during batch fluxing of 4500 lbs melt of 99.85% Al. Comparison between chlorine and nitrogen-10% Freon 12 .

		Emission Rate (lb/hr)						
	Flow* Rate	Solids					Gases	
Fluxing Gas	scfm	Al_2O_3	$AlCl_3$	AlF_3	Others**	Total	HCl	HF
Cl_2	3.2	21.6	0.4	0.01	8.8	30.8	42.3	0.01
N_2-10% Freon 12	3.2	3.2	0.1	0.03	0.9	4.2	4.7	0.03

* Flow rate required to degas to 0.07 cc H_2/100g Al in 30 min.

** Ignition losses and unanalyzed oxides.

Plant Tests of Inert Gas-Freon Melt Treatment

Routine production tests were carried out on 45,000 lb melts, in a gas fired open hearth furnace equipped with a flux bay with 10 graphite tubes (1/2 in.diameter I.D.) lowered through the roof, allowing for continuous fluxing during casting.

The test program consisted in a comparison between fluxing with pure chlorine and nitrogen-5% Freon 12 . The alloys included 1145, a Mg-free alloy and 3004, a Mg-containing alloy, and the emission rates are shown in Table 2.

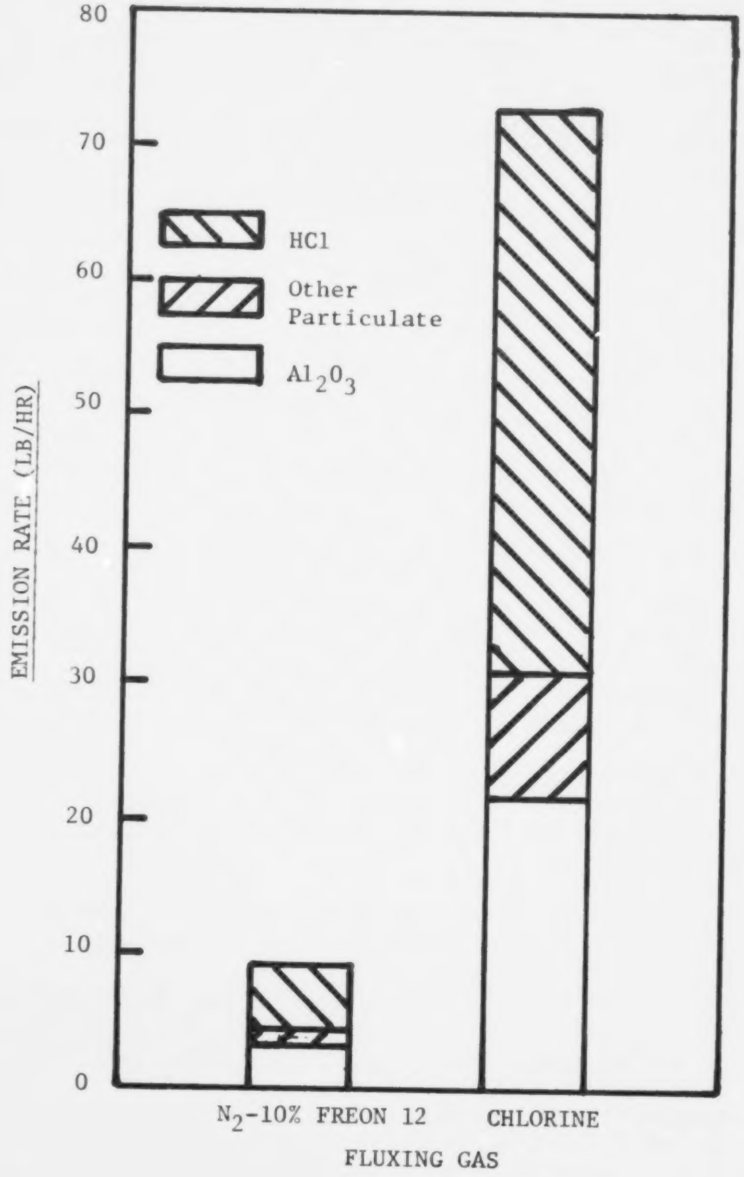

FIGURE 1 - Emission rates during batch fluxing of 4500 lbs melt of 99.85% aluminum. Comparison between chlorine and nitrogen-10% Freon 12 at 90 l/min (3.2 scfm) flow rate.

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Table 2 - Emission rates during continuous fluxing of 45,000 lbs melts of alloys 1145 (Mg-free) and 3004 (Mg-bearing). Comparison between chlorine and nitrogen-5% Freon 12 .

				Emission rate (lb/hr)						
				Solids					Gases	
Alloy	Fluxing Gas	Flow* Rate (scfm)	Cast No.	Al_2O_3	$AlCl_3$	AlF_3	Others**	Total	HCl	HF
1145	Cl_2	4.5	1	16.50	0.05	0.00	3.45	20.00	42.30	0.10
			2	16.60	0.58	0.00	0.92	18.10	41.90	0.04
	Nitrogen-5% Freon 12	3.2	1	2.26	0.01	0.02	1.31	3.60	6.00	0.26
			2	2.00	0.01	0.01	0.48	2.50	5.40	0.28
3004	Cl_2	5.5	1	1.37	0.14	0.00	0.31	1.82	54.50	0.07
	Nitrogen-5% Freon 12	3.2	1	0.07	0.02	0.01	0.44	0.54	3.20	0.14
			2	0.02	0.01	0.01	0.20	0.24	2.00	0.40

*Flow rates were adjusted to give equivalent degassing for both fluxing gases, down to 0.06-0.09 ccH_2/100g Al.

**Ignition losses and unanalyzed oxides.

At the time of the test, the proposed state regulation called for a maximum solid particulate emission rate of 20 lb/hr for the entire plant. As shown in Table 2, this cannot be met with Cl_2 for Mg-free alloys if more than one casting unit is operating at one time.

The Mg-containing alloys could be fluxed with Cl_2 even if all 8 furnaces were operating at once, on the basis of solid particulate emission. However, the HCl emission rate for both Mg-free and Mg-bearing alloys are so high with Cl_2 as to rule out its use.

Conversely, the nitrogen-5% Freon 12 clearly appears capable of meeting regulations as well as efficiency requirements for hydrogen removal.

Additionally, the stack gas sampling indicated that the phosgene ($COCl_2$) content was below 0.1 ppm and the CO content was below 10 ppm, detection limits of the analytical method respectively, and OSHA standards as well.

Because nitrogen-Freon 12 mixtures release N, C, F and Cl, there was a theoretical possibility that C and N might react to form cyanide (CN) which would be likely to be in the dross since this is where the carbon is present. This was not expected to be a real problem since the earlier 4500 lb melt fluxing had resulted in less than 1% carbon in the particulate collected during fluxing with a nitrogen-30% Freon 12 mixture. This was confirmed by the actual level of CN found in the dross which was only in the 0.5-4.5 ppm range, the range in the reagent blank considered free of cyanide.

Visible Emission:

An idea of the limitation placed on visible emission can be obtained from the following quote from the Visible Emission regulations of the State of Tennessee: "No discharge of a visible emission ... with a density greater than number one (1) on the Ringelmann chart or an opacity in excess of twenty (20) percent for an aggregate of more than five (5) minutes in any one (1) hour or more than twenty (20) minutes in any twenty-four (24) hour period."

Visible emissions tending to produce a black plume are evaluated in terms of the Ringelmann scale, while visible emissions tending to produce a non-black plume are evaluated in terms of equivalent opacity and expressed as percent opacity. The major emission problems are encountered with Mg-free alloys; therefore, the potential of inert gas-freon treatment in reducing emission levels was initially tested with such alloys.

Table 3 - Comparison between visible emissions during batch fluxing of 25,000 lbs melts of Mg-free aluminum alloys.

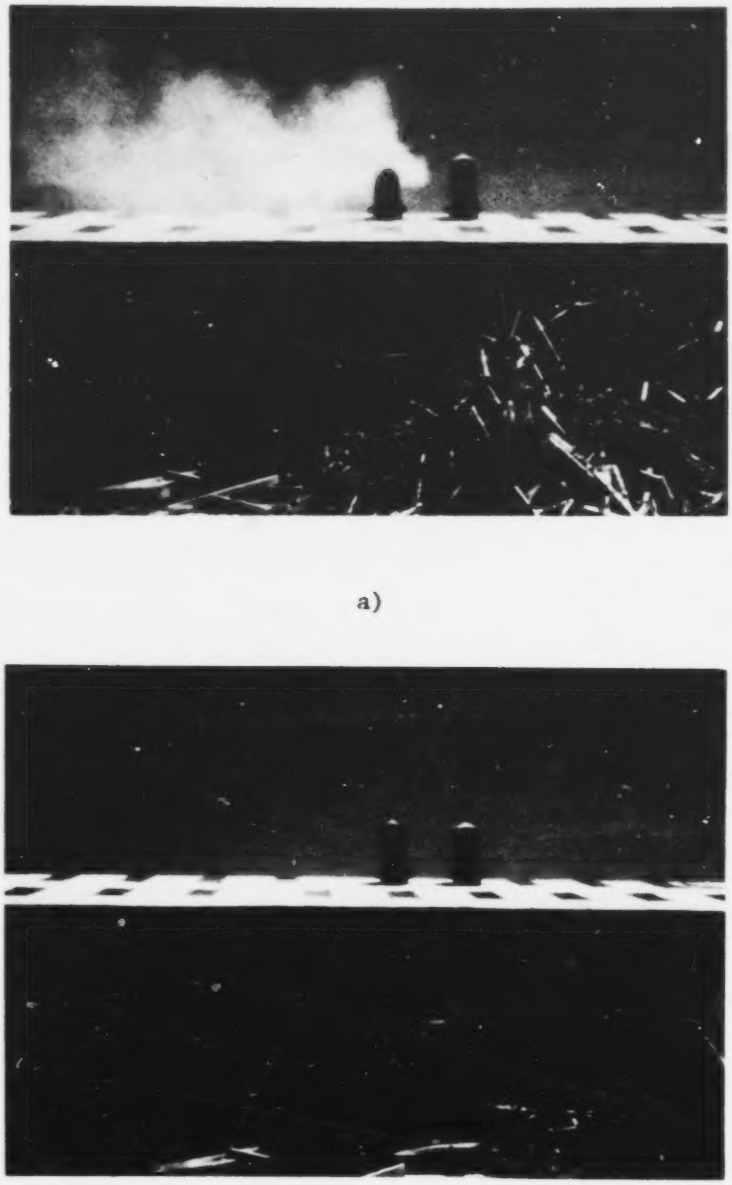
Alloy	Fluxing Gas	Flow Rate (scfm)	Flux*	Visible emission** (Ringelmann No.)
2011	Cl_2	6	None	5
	Cl_2	4	None	4 to 5
	Cl_2	4	A	5
	N_2-5% Freon 12	6	A	1/4 - 1/2
	N_2-3% Freon 12	6	B	1/4
4043	Cl_2	6	None	5
	Cl_2	4	None	4 to 5
	N_2-3% Freon 12	6	B	1/4

*Flux A (demagging flux): 50% AlF_3, 24% NaCl, 24% KCl, 2% K_3AlF_6
Flux B: 48% NaCl, 48% KCl, 4% K_3AlF_6

**Maximum allowed is Ringelmann No. 1

As summarized in Table 3, it is apparent that fluxing Mg-free alloys with nitrogen-3 to 5% Freon 12 results in visible emissions well below the required Ringelmann No. 1, whereas Ringelmann numbers in the 4-5 range are observed when using pure chlorine. Figure 2 dramatically illustrates this point.

Another type of emission problem results during dross treatment of alloys with a salt flux. For example, dross treatment of a Mg-free alloy (3003) containing 25% painted scrap and using a commercial flux (55% NaCl, 15% KCl, 15% Na_2SiF_6, 7% Na_2SO_4, 7% $NaNO_3$) created visible emissions above the specified 20% opacity level during the first 5 minutes of the treatment, still



a)

b)

FIGURE 2 - Typical visible emission during fluxing of a Mg-free aluminum alloy melt.
a) Chlorine fluxing - Ringelmann 5
b) Nitrogen-5% Freon 12 - Ringelmann 1/4-1/2

staying above 10% for the next 15 minutes. When switching to N_2-3% Freon 12 at 4 scfm, visible emissions never exceeded 5% opacity. To get equally low metal losses from the dross, it was however found necessary to increase the Freon 12 content from 3 to 10%. Even though no measurement was made to confirm this point, it was expected that the resulting opacity of the plume would not exceed 15%.

As indicated earlier, chlorine fluxing of Mg-bearing alloys poses no real visible emission problems. However, some tests were run to confirm the expectation that inert gas-Freon 12 melt treatment would do as well, if not better, than chlorine. For example, chlorine fluxing at 8 scfm for a clean melt of Mg-bearing alloy (5056 alloy) was found acceptable in terms of visible emission, with a Ringelmann number of 1/2. However, nitrogen-3% Freon 12 with a 6 scfm flow rate resulted in still slightly lower Ringelmann numbers, around 1/4. In the case of 6063 alloy, stack emissions were also about 10% opacity or 1/2 on the Ringelmann scale, when fluxing with N_2-3% Freon 12 at a total flow rate of 6 scfm with a 48% NaCl, 48% KCl and 4% K_3AlF_6 flux.

Technical Performance of Inert Gas-Freon Process

The success of the inert gas-Freon process in meeting environmental requirements has been amply demonstrated above. It is important to emphasize that this has been achieved at no sacrifice in the quality of the aluminum melt. A few examples are reported below.

Hydrogen removal:

A low hydrogen level in the melt is required to give ingots free of porosity. A series of degassing tests were run under essentially identical conditions to compare chlorine and N_2-10% CO-10% Cl_2 to nitrogen-5% Freon 12. Hydrogen levels were measured in the transfer trough between melting and holding furnace, using the initial bubble method (FMA hydrogen tests).

Table 4 - Typical hydrogen levels in plant fluxing tests - 45,000 lbs melts.

	Hydrogen level (cc/100g Al)		
	Alloy		
Fluxing Gas	1100/1145	3004	5052
N_2-5% Freon 12	0.08	0.07	0.08
Cl_2	0.08	0.09	0.07
N_2-10% CO-10% Cl_2	0.11	--	0.11

The typical results in Table 4 illustrate for several alloys (1100, 1145, 3004, 5052) the fact that nitrogen-5% Freon 12 is as effective as pure chlorine, and somewhat more effective than N_2-10% CO-10% Cl_2 in terms of hydrogen removal.

A continuous production evaluation over a 3 month period demonstrated the consistency of these results as shown in Table 5.

Table 5 - Distribution of hydrogen levels during 3 month plant production. 45,000 lbs melts. 1XXX, 3XXX and 5XXX series alloys.

Fluxing gas	Total	Number of casts per hydrogen level					
		<0.10cc/100gAl		0.11-0.15cc/100gAl		>0.15cc/100gAl	
N_2-3% Freon 12	50	34	(68%)	15	(30%)	1	(2%)
Cl_2	83	59	(71%)	22	(27%)	2	(2%)

Melt cleanliness:

Melt cleanliness, that is the amount of non-metallic particulate in the melt, can be assessed directly by means of a pressure filter test. Approximately 20 lbs of melt is ladled into a preheated crucible and then forced under air pressure through a fine porous filter disc which catches all the non-metallics. After sectioning and metallographic examination of the disc and the adjacent residual metals, melt cleanliness is rated by the thickness of the band of non-metallics.

Table 6 - Melt cleanliness as a function of fluxing gas and alloys, as measured by pressure filter test of continuously fluxed metal - 45,000 lbs melts.

Alloy	Fluxing Gas	Average Non-metallic Band Thickness (mm)
1100, 1145 (Mg-free)	N_2-5% Freon 12	0.13
	Cl_2	0.35
	N_2-10% CO-10% Cl_2	0.13
3004, 5052 (Mg-bearing)	N_2-5% Freon 12	0.11
	Cl_2	0.11
	N_2-10% CO-10% Cl_2	0.50

The test results shown in Table 6 demonstrate that whether for Mg-free or Mg-bearing alloys, N_2-5% Freon 12 is equivalent to the best gas mixture, which is N_2-10% CO-10% Cl_2 for Mg-free alloys and pure Cl_2 for Mg-bearing alloys.

Later production control tests for 5252 alloy over a 3 month period related the cleanliness of the final sheet material to the nature of the fluxing gas mixture. In this case, the number of linear defects in a 10in x 10in sample was measured and reported in Table 7.

Table 7 - Metal cleanliness as a function of fluxing gas - 3 month plant production - 5252 alloy

Fluxing Gas	Number of Samples	Total Number of Defects*	Average Defect Per Sample
N_2-3% Freon 12	73	129	1.8
Cl_2	51	163	3.2

* Type II linear defects in 10in x 10in sample.

The use of N_2-3% Freon 12 resulted in an average of 1.8 defect/sample, whereas with Cl_2, this number was 3.2, or 80% higher.

Denatrification:

Sodium levels below about 5 ppm are desirable in order to improve castability and hot workability of high Mg, 5XXX series alloys. In tests with 30,000 lbs melts of 5056 alloy, it was demonstrated that N_2-3% Freon 12 is as effective as Cl_2 in removing Na provided a Na-free cover flux (55% KCl, 42% $MgCl_2$ and 3% CaF_2) is used. Detailed results are presented in Table 8 for one cast with Cl_2 and two casts with N_2-3% Freon 12 .

Table 8 - Denatrification of 30,000 lb melt of 5056 alloy.

Fluxing Gas	Cast No.	Na level (%) Initial	Na level (%) Final	Fluxing Gas Usage (scf)	Time (min)
N_2-3% Freon 12	1	0.0010	<0.0007*	120	65
	2	0.0020	<0.0007*	120	150
Cl_2	1	0.0014	<0.0007*	160	70

* Below detection limit.

Melt losses:

For purely economic reasons and to minimize the amount of material which must eventually be disposed of after reclamation of the metal content, it is important that melt losses through dross generation be minimized.

Based on plant scale tests, it was determined that for Mg-free alloys, melt losses with N_2-3% Freon 12 were equivalent to N_2-10% CO-10% Cl_2 and slightly higher than Cl_2. For Mg-bearing alloys, however, N_2-3% Freon 12 resulted in the lowest melt losses, Cl_2 and N_2-10% CO-10% Cl_2 giving respectively higher losses.

Therefore, depending on the exact product mix between Mg-free and Mg-bearing alloys, the net melt losses with N_2-3% Freon 12 are expected to be equivalent to Cl_2 and better than N_2-10% CO-10% Cl_2.

Conclusion

The inert gas-Freon 12 melt treatment process has been demonstrated, up to large scale industrial production condition, to be technically equivalent to pure chlorire treatment in all important aspects including hydrogen removal, non-metallic inclusions control and denatrification. At the same time, it solves all the environmental and occupational safety problems caused by chlorine. The major favorable aspects are the non-toxicity of the gas mixture itself and the substantial reduction, well below existing standards, of both air pollution and visible emissions levels.

This process is now used exclusively for melt treatment in the Consolidated Aluminum Corporation rolling mill at Hannibal, Ohio.

ACKNOWLEDGEMENTS

The authors wish to acknowledge the extensive contributions made by P. E. Sevier and J. C. Yarwood of Olin Metals Research Laboratories to the development and plant implementation of the inert gas-Freon 12 melt treatment process. The cooperation of the personnel of the various Consolidated Aluminum Corporation cast shops during plant trials was also invaluable.

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COPPER INDUSTRY

Session Chairman: Ivor E. Campbell
Clyde Williams and Company
Columbus, Ohio

INCO'S OXYGEN FLASH SMELTING PROCESS FOR COPPER AND NICKEL CONCENTRATES OFF-GAS HANDLING AND IMPURITY DISTRIBUTIONS

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ABSTRACT

The Copper Cliff Smelter of the Inco Metals Company has been autogenously smelting concentrates in an oxygen flash furnace of Inco design since 1952. Over the past twenty-seven years, some seven million tonnes of copper concentrates have been treated using this technique. During a recent demonstration run, some 11,000 tonnes of nickel concentrates were also smelted using the same process. The nominal capacity of the present furnace is 1400 tonnes of concentrate per day, but throughputs as low as 600 - and as high as 1650 tonnes per day have been achieved while remaining in an autogenous mode of operation.

Two main gas streams are produced in this process. The two top-discharged fluid bed reactors, in which the furnace feed is dried, average 650 m^3/min (24,300 scfm) of a gas-solid mixture at 105°C. The dried feed is separated in product baghouses which operate at collection efficiencies of 99.9%.

The off-gases from the furnace proper amount to only 115 m^3/min (4300 scfm) at the current throughput of ∿1100 tonnes per day. They contain less than 3% of the feed as dust and between 25 to 70% of the bismuth, 20 to 50% of the lead and arsenic, and 5 to 15% of the selenium in the concentrates smelted, depending on type of concentrate and oxygen potential. About 85% of these gases are scrubbed and sent, at 70 to 80% SO_2 by volume, to a liquefaction plant averaging ∿270 tonnes of liquid SO_2 per operating day. The dust and impurity collection efficiency of the furnace gas handling system is 99.9+%.

The impact that incorporation of the Inco oxygen flash smelting process would have on dust, impurity and SO_2 containment in an integrated smelter is also discussed.

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THE COPPER CLIFF SMELTER

In a typical year, the Inco Metals Company mines, and processes, at its Copper Cliff complex, some 15 million tonnes of ore from which the main products are some 160,000 tonnes of nickel, 140,000 tonnes of copper, 600,000 tonnes of iron ore, 80,000 tonnes of liquid sulfur dioxide and 800,000 tonnes of sulfuric acid. Significant amounts of cobalt, precious metals, selenium and tellurium are also recovered.

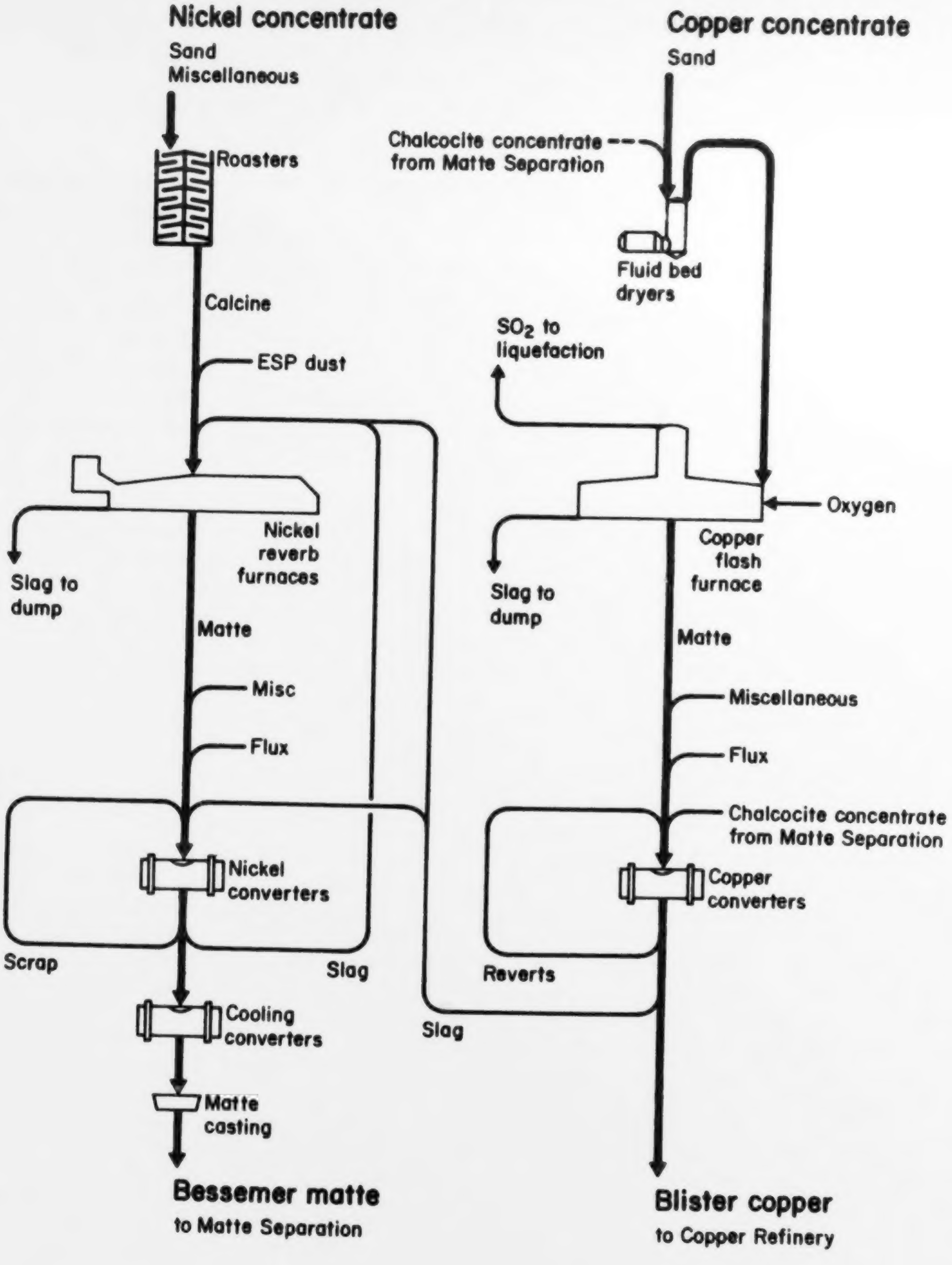
At the heart of the pyrometallurgical activities are the nickel and copper circuits of the Copper Cliff Smelter through which some 1.6 million tonnes of "nickel" concentrates and 0.4 million tonnes of chalcopyrite concentrate are treated each year. Copper reporting in the "nickel" concentrates (about 25% of copper input) and nickel reporting in copper concentrate (only 3% of nickel input) cause substantial intercircuit metal transfers, as the current smelter flowsheet shown in Figure 1 illustrates. The reasons for this interdependence merit further description, particularly since it influences directly the mode of operation of the oxygen flash furnace.

The "nickel" concentrates, averaging 10% Ni and 2% Cu, are partially calcined in multi-hearth roasters, smelted in reverberatory furnaces and blown in Peirce-Smith converters to produce a sulfur deficient nickel-copper-cobalt sulfide phase called Bessemer matte. This Bessemer matte is cast into 22 tonne molds, cooled at a controlled rate and then separated in the Matte Processing Plant to produce heazlewoodite, metallic and chalcocite concentrates. The heazlewoodite concentrate is treated in fluid bed roasters to produce various grades of nickel oxide products while the metallic fraction becomes the feed to the Copper Cliff Nickel Refinery. The chalcocite concentrate, contaminated with ~5-6% Ni as heazlewoodite (Ni_3S_2) is blown to blister in the copper circuit converters.

The chalcopyrite concentrate, analyzing about 30% Cu and 1% Ni, is dried in fluid bed reactors, smelted in the oxygen flash furnace and blown to blister copper in Peirce-Smith converters. The blister copper composition is controlled at ~98% Cu and <0.7% Ni (the rest being dissolved oxygen) and most of the nickel entering the copper circuit is thus concentrated in the slag raised during the copper converters' slag blows and the dry mush raised during the finishing blows. Consequently, the only way of controlling the nickel levels in the copper circuit is to return the copper converter slag and mush to the nickel circuit.

The fact that no copper converter slag is returned to the copper flash furnace is thus a response to the nickel bleed requirements and not an intrinsic short-coming of the furnace itself. As a matter of fact, recent tests (Dec. 1977-Jan/Feb. 1978) have indicated that recycle of copper converter slag to

FIGURE I COPPER CLIFF SMELTER FLOWSHEET


Nickel concentrate
Sand
Miscellaneous
Roasters
Calcine
ESP dust
Nickel reverb furnaces
Slag to dump
Matte
Misc
Flux
Nickel converters
Scrap
Slag
Cooling converters
Matte casting
Bessemer matte
to Matte Separation
Copper concentrate
Sand
Chalcocite concentrate from Matte Separation
Fluid bed dryers
SO2 to liquefaction
Oxygen
Copper flash furnace
Slag to dump
Matte
Miscellaneous
Flux
Chalcocite concentrate from Matte Separation
Copper converters
Reverts
Slag
Blister copper
to Copper Refinery

the flash furnace is possible with very little disruption of normal operation. A permanent slag return launder is consequently being installed on the furnace.

OXYGEN FLASH SMELTING PROCESS

The oxygen flash smelting process was developed in the 1940's by the then International Nickel Company of Canada to treat either copper or nickel concentrates, as the earliest references to this work indicate[1],[2]. The first commercial unit was built in 1952 to handle 450 tonnes per day (tpd) of concentrates. The current furnace dates from 1968 and has a nominal capacity of ∿1400 tpd.

Over the past twenty-seven years, some seven million tonnes of copper concentrates have been treated by the oxygen flash smelting process. The process was never commercially used on nickel concentrates for two main reasons: a) the Copper Cliff Smelter was extensively enlarged and modernized in the ten years prior to 1940[3] and b) further improvements to the existing nickel circuit[4],[5], particularly extensive use of tonnage oxygen[6], considerably improved its efficiency. In 1976 however, a nickel concentrate campaign was carried out in the commercial unit, during which some 11,000 tonnes of various nickel concentrates were smelted over ten days of operation. The process was found to handle nickel feed with the same ease as it handles copper concentrate[7].

The oxygen flash smelting process, its evolution and current practice have been extensively discussed in the literature ([8-14]). Figure 2 illustrates in some detail the arrangement of the furnace and its ancillary equipment. Typical operating conditions are quoted in Tables 1 and 2, the reader being referred to the literature for description of the parts of the process beyond the scope of the present paper.

Fluid Bed Dryers and Baghouses

Prior to 1971, the feed to the furnace was dried in small rotary kilns, 1.2 m in diameter and 5.5 m long, that were originally fired with pulverized coal[8] and later with natural gas[10]. Five such kilns were used, four on concentrates and one on sand flux. The kilns were "air-swept" to pick-up, transport and elevate the dried products to the furnace feed bins[9]. The dried concentrate and sand were separated from the gas stream in a series of dry cyclones.

The total drying capacity of the concentrate kilns was only 950 tpd, well below the capacity of the flash furnace itself. Moreover fugitive emissions from the kilns and associated multiclones were difficult to control and periodically caused unacceptable conditions in the workplace.

FIGURE 2 FLASH FURNACE GENERAL ARRANGEMENT

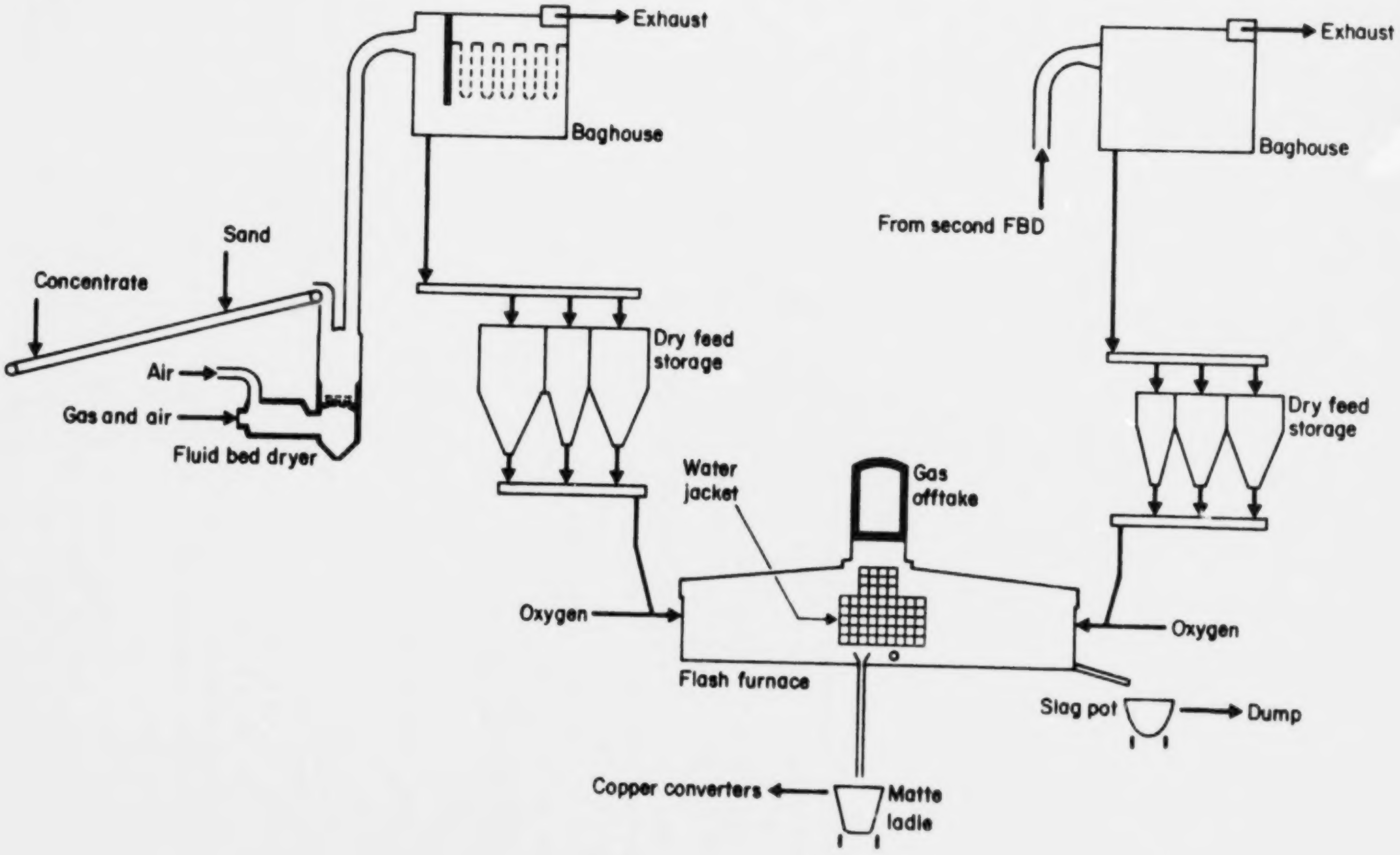

Exhaust
Baghouse
Sand
Concentrate
Air
Gas and air
Fluid bed dryer
Dry feed storage
Water jacket
Gas offtake
Oxygen
Flash furnace
Copper converters
Matte ladle
Exhaust
Baghouse
From second FBD
Dry feed storage
Oxygen
Slag pot
Dump

Table 1. INCO'S OXYGEN FLASH FURNACE

1977 OPERATING DATA FOR SMELTING OF COPPER CONCENTRATE

	Dry	Composition, Wt. %				
Inputs	Tonnes/Day	Cu	Ni+Co	Fe	SiO_2	S
Concentrate	1,101	30.1	1.2	31.4	1.7	33.5
Cu Converter Dust	28	41.8	3.4	8.0	2.5	25.0
Flux	96	-	-	2.2	77.8	-
O_2	219	-	-	-	-	-
TOTAL	1,444					
Outputs						
Matte	794	42.1	1.7	29.2	-	24.6
Slag	289	0.6	0.1	40.9	32.4	1.4
Scrubber Sludge*	17	42.0	1.8	20.0	2.8	30.6
SO_2	343	-	-	-	-	50.0
TOTAL	1,443					

* Before neutralization

Table 2. INCO'S OXYGEN FLASH FURNACE

1976 TEST OPERATING DATA FOR SMELTING OF NICKEL CONCENTRATE

	Dry Tonnes/Day	Composition, Wt. %					
Inputs		Cu	Ni	Co	Fe	SiO_2	S
Concentrate + Flux	1,232	2.6	8.0	0.29	34.1	20.2	25.0
Recycles*	36	24.3	5.8	0.25	20.0	10.0	15.0
O_2	294	-	-	-	-	-	-
Residual Cu Matte**	9	45.1	3.2	0.10	25.3	-	24.5
TOTAL	1,571						
Outputs							
Matte	435	9.7	23.2	0.58	35.2	-	27.1
Slag	659	0.18	0.34	0.14	36.6	38.0	1.0
Scrubber Sludge***	37	2.9	4.4	0.24	26.8	2.4	19.5
SO_2	405	-	-	-	-	-	50.0
TOTAL	1,536						

* Contain dust from copper converter operations.

** Cu matte washed out of furnace during nickel campaign.

*** Before neutralization, including sulfates in solution.

Consequently these driers were replaced in 1971 by two fluidized bed drying lines each capable of handling 900 tpd of a proportioned concentrate-sand mixture. Each drying line (Figure 2) consists of a 210,000 kJ/min combustion chamber fired with natural gas or light industrial oil, a 335 kW blower capable of delivering 670 m^3/min of dilution air at 19 kilopascals relative pressure (kPa), a refractory lined fluid bed vessel and a collection baghouse.

The wet concentrate-flux mixture (at ∿8-10% H_2O) is introduced to the dryer through a rotary sealing feeder of Inco design. The solid particles are suspended in the upward stream of hot combustion gas (the gas bed velocity is about 3.5 m/s at 10-14 kPa) and are thus dried to 0.1-0.3% free H_2O. The moist gases and dry solids are drawn out through the dryer roof for separation in the product baghouse. Normal operating conditions are:

- a wind box temperature of 315°C
- a bed temperature of 120°C

and - an off-gas volume of 650 m^3/min at ∿10% moisture and 105°C.

The product baghouses, are 7.6 m high (including hoppers) by 9.8 m long and 3.7 m wide. Each contains 672 bags, 11.5 cm in diameter by 2.4 m in length, which are suspended vertically and supported internally by wire-frame retainers. A tube sheet supports the bags which are clamped onto venturis. Each baghouse has two sides, each side containing five banks. Purging by reverse pulse air is sequential and accomplished automatically. Various filtering media were tested, the current bags being Nomex felt fabric. The exhaust fans are driven by 75 kW, 650 rpm motors and are rated at ∿700 m^3/min. Normal operating conditions are:

- an inlet loading of ∿900 g/m^3
- an inlet temperature of 105°C
- an outlet loading of ∿0.7 g/m^3 (0.2 g/m^3 with new bags)
- an outlet temperature of ∿80 to 100°C

and - and outlet pressure of -2 to -2.7 kPa (baghouse differential varies between 0.5 and 2.0 kPa, i.e. 2" and 9" H_2O).

The product baghouses thus clean some 650 m^3/min of gas at better than 99.9% collection efficiency.

The baghouse off-gases are piped to the copper converter flue and are further treated in a dry electrostatic precipitator (ESP) that handles approximately 27,000 m^3/min of gas at 150°C. The metal collection efficiency of this ESP is currently ∿92%, the dust recovered being recycled to the flash furnace. The fluid bed dryer off-gases are therefore de-dusted with an overall efficiency of >99.99%, the dust loadings being brought down to <0.06 g/m^3 (<0.026 grain/scf) before the gases are ducted to the 380 metre chimney.

The overall availability of each drying line is better than 98%. Because of limiting factors beyond the furnace itself, operating time is only 60 to 80%.

Flash Furnace and Off-Gas Handling System

The oxygen flash furnace is fired from both ends by four burners, two at each end. The solids are withdrawn from individual burner feed bins at a controlled rate and transferred to the burners by screw conveyors and gravity, the screw conveyors being vented to the copper converter flue. Recycled dust and touch-up sand, if required, are added to the screw conveyors, also at controlled rates.

At the burners, the feed is suspended in a horizontal flow of oxygen (>96% purity, ∿105 kPa) and injected into the furnace. The oxygen readily reacts with part of the sulfur and iron in the feed to form SO_2 and iron oxides. Silica, contained in the flux and concentrate, combines with the iron oxides to form a fayalite slag. Copper, nickel and cobalt and the remaining iron and sulfur collect in the matte.

The main oxidation reactions are:

Copper Concentrate

$$2CuFeS_2 + 2.575\ O_2 = Cu_2S + FeS + 0.55\ FeO + 0.15\ Fe_3O_4 + 2\ SO_2$$

Nickel Concentrate

$$2FeS_{1.14} + 1.855\ O_2 = FeS + 0.55\ FeO + 0.15\ Fe_3O_4 + 1.28\ SO_2$$

Hence, about 1.55 and 1.38 tonnes, respectively, of SO_2 are produced per tonne of O_2 used while flash smelting copper or nickel concentrates. These figures translate to 1.50 and 1.33 tonnes, respectively, of SO_2 per tonne of commercial oxygen used at 96% purity. Oxygen utilization is 100%.

The flame gases analyze 94.5% SO_2, 1% N_2, 1.7% CO_2 (from combustion of flotation reagents and wood debris in the concentrates) and 2.8% Ar (argon from the tonnage oxygen production process) before dilution by inleakage to the furnace. In the furnace proper, this inleakage occurs mostly around the burners and is estimated at about 16 m^3/min. The oxygen in the inleakage is known to react with the feed; the furnace off-gases therefore assay 83% SO_2, 13% N_2, 1.5% CO_2, 2.5% Ar and <0.02% O_2 when they leave the furnace. They also carry 2-3% of the feed as dust during copper concentrate smelting (∿3-4% of nickel concentrates which are finer) and significant fractions of the impurities contained in the feed.

Notes: 10 m^3/min at 0°C = 373.2 scfm at 60°F
1 kPa relative pressure = 4" H_2O or 0.15 psig

At a typical production rate of 1,100 tpd of concentrate, the off-gases have a volume of only 104 m^3/min and exit the furnace at 1260°C. These gases are thoroughly cleaned prior to delivery to an SO_2 liquefaction plant operated by Canadian Industries Ltd. (CIL)[15].

The furnace gas handling system, shown in Figure 3, was originally built in 1952-53[16]. This system has been renovated from time to time and somewhat improved, but the current design is basically the same as the original. The main units are:

- a 9.4 m high (including hoppers) by 4.9 m wide by 18.3 m long settling and radiation chamber originally constructed of mild steel, but currently made of 316 stainless. About one third to one half of the entrained dust is collected in this chamber (and pneumatically returned to the furnace) while the gases are cooled to 700°C at the same time.

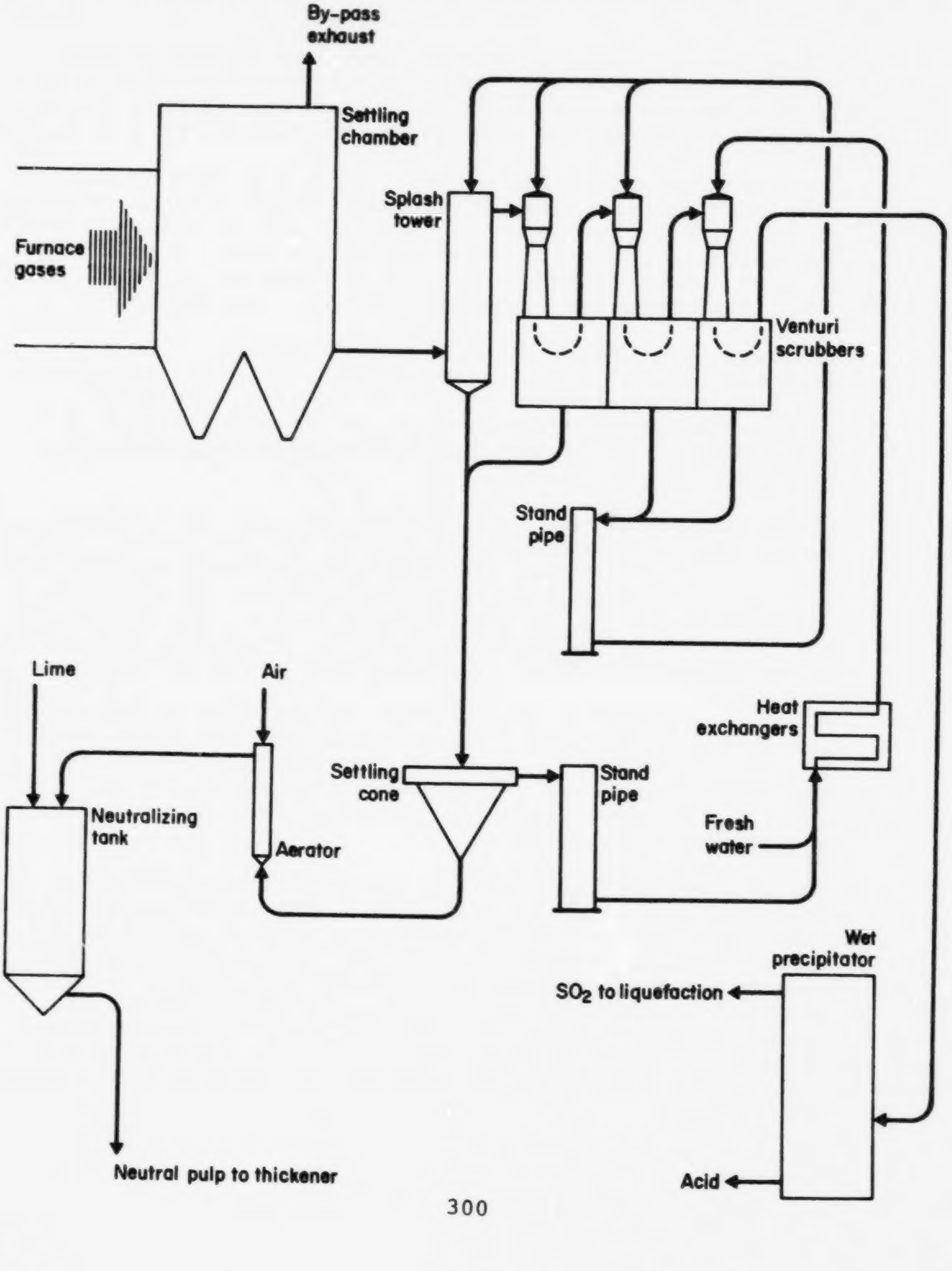
- a splash tower which is an unpacked chamber 1.2 m in diameter and 5.5 m high. The chamber and its two spray nozzles are constructed of 316 stainless. The gases rise in the chamber countercurrently to the water sprays and are cooled to ∿60°C in the process; scrubbing efficiency is ∿93%.

- Three venturi scrubbers in series, mounted on a common tank. Again construction is all of 316 stainless. The common tank is divided into three sections by baffles in such a way that gases cannot bypass any unit, the spray water accumulating to form a water seal at the bottom of the tank. Flow in these venturis is cocurrent, the gases being cooled to ∿32°C; scrubbing efficiency is ∿95%.

and - a wet electrostatic precipitator constructed of antimonial lead and acid resistant brick. This wet ESP is protected from any sudden increase in draft, which could collapse the lead tubes, by a water seal connected to the unit near its inlet. Gases leave the ESP at 25-35°C. Collecting efficiency in this ESP is ∿97%.

Draft through the gas handling system is provided by one of two centrifugal fans rated at 120 m^3/min. These fans have double inlets, the impellers being constructed of a cast spider with welded blades. Again construction is all 316 stainless. The gases leaving the fan, compressed to 25 kPa and heated to 65°C in the process, are carried to the CIL liquefaction plant in a 0.5 m Ø pipeline constructed of fiberglass-reinforced-polyester wrapped polyvinyl-chloride (FRP/PVC). The line is covered with 5 cm of fiberglass insulation to prevent heat loss and condensation within the pipe.

FIGURE 3 FLASH FURNACE GAS CLEANING SYSTEM


By-pass exhaust
Settling chamber
Furnace gases
Splash tower
Venturi scrubbers
Stand pipe
Lime
Air
Heat exchangers
Settling cone
Stand pipe
Neutralizing tank
Aerator
Fresh water
Wet precipitator
SO2 to liquefaction
Neutral pulp to thickener
Acid

The scrubbing system operates under draft which varies from -0.12 kPa (0.5" H_2O) in the settling chamber to -3 kPa in the wet ESP. Because of this vacuum, and in spite of all efforts to completely seal the settling chamber, inleakage in this unit occurs: an estimated 11 m^3/min of air enters the chamber diluting the furnace off-gases from 83% SO_2 to about 75% SO_2 (mixed with 20% N_2, 2% Ar, 1% CO_2 and <1% O_2) and increasing their volume from 104 m^3/min to 115 m^3/min. No further dilution occurs in the wet scrubbing system itself.

Scrubbing water flows are shown in Figure 3. The underflow from the splash tower and the first venturi scrubber, about 1200 liters per minute, is taken to a 316 stainless steel cone thickener for settling. The settling cone overflow, with make-up fresh water, is cooled from 55°C to 20°C through a series of shell and tube heat exchangers, also constructed of 316 stainless. The cooled water is recycled to the third venturi scrubber.

The underflow from the settling cone, ∿60 ℓ/min at 20% solids, is stripped of SO_2 by air purging at a rate of 3 m^3/min, neutralized with lime and reverted to the concentrator nickel circuit.

The underflow from the second and third venturi scrubbers, ∿1500 ℓ/min, is recycled to the splash tower and the first two scrubbers.

Overall scrubbing efficiency is ∿99.99% with less than 2 kg/day of dust being collected on the liquefaction plant filters (see Table 3). For several reasons, one being the need to increase draft on the furnace when changing burners, another the limited capacity of the scrubbing system and the liquid SO_2 plant, only about 85% of the flash furnace gases are handled through the system described. The remaining 15% are exhausted from the settling chamber through a bell damper from which they are vented to the converter flue which leads to the dry ESP mentioned in the previous section. Since the efficiency of CIL's liquefaction plant is 95%, liquid SO_2 production averages 270 tpd out of the 340 tonnes of SO_2 produced while the furnace is in normal copper service.

Impurity Elimination

The Copper Cliff concentrates are, by most standards, relatively free of impurities. The copper concentrates generally average less than 0.01% Bi + As + Se and less than 0.10% Pb. The nickel concentrates assay less than 0.01% Bi+Se and less than 0.04% Pb+As, these four impurities being the only ones that have traditionally been followed.

For this reason and others, such as smelting and converting oxygen potentials or amount and point of re-entry of various

Table 3. FLASH FURNACE GAS HANDLING SYSTEM

COLLECTION EFFICIENCIES

Collected after passage through	Cumulative, %								
	Weight	Cu	Ni+Co	Fe	As	Zn	Se	Pb	Bi
Settling chamber	42	37	49	52	38	46	25	18	19
Splash tower	96	95	96	97	96	96	94	93	94
Venturi scrubbers	99.8	99.8	99.8	99.9	99.8	99.9	99.8	99.7	99.7
Wet Cottrell	99.995	99.999	99.985	99.999	99.995	99.998	99.991	99.989	99.976

recycle streams, the impurity distributions obtained in our circuits, particularly in the oxygen flash furnace, are representative only of our own operation.

Distribution of the four impurities mentioned between matte, slag and scrubber slurry are given in Figures 4 to 7. Data are quoted for nickel and copper flash smelting without slag return as well as for conventional nickel roaster+reverb smelting and copper flash smelting with slag return. The data are plotted versus the oxygen potential sustained by the slags under the various operating conditions (calculated from the Fe^{3+}/Fe^{2+} ratio in the slags).

As shown, about 20 to 30% of Bi, Pb and As and only 3% of Se are recovered in the scrubber slurry under normal copper flash furnace operation (i.e. without slag return). In normal nickel service, these fractions are greater amounting to 50-70% for Bi, 30-50% for Pb, ∿40% for As and 7-15% for Se depending on the oxygen potential.

As shown in Table 3, these impurities are collected in the gas handling system in somewhat different manners depending on their physical state. As and Zn behave in much the same fashion as Cu, Ni, Co and Fe: 40 to 50% of the total amount of these elements carried by the gas stream are recovered in the settling chamber. This behaviour indicates that As and Zn are already in a solid state when they enter the settling chamber or that, at the very least, they condense early enough in the settling chamber to behave as Cu, Ni, Co and Fe (settling chamber temperatures drop from 1300°C to 700°C). On the other hand, only 20-25% of the Se, Pb and Bi carried by the gas stream are recovered in the settling chamber. A fraction of these impurities could thus still be in a volatile state at the settling chamber outlet temperature.

In any case, about 99.99% of the various elements are scrubbed out in the remainder of the gas handling system. The neutralized settling cone slurry is then recycled to the reverberatory furnace circuit where the impurities are distributed mostly between slag and matte (see Figures 4 to 7).

EMISSION CONTROL USING OXYGEN FLASH SMELTING PROCESS

The nickel and copper circuit interdependence, and the mode of operation of the oxygen flash furnace described in the previous sections are unique to Inco Metals and have grown out of the difficulties encountered in the treatment of the Sudbury ores. This, and the previous reluctance of the company to market its technology, probably account for the fact that only two oxygen flash smelting furnaces are in operation at present (Inco's and an identical unit located in Almalyk, USSR) in spite of the fact

FIGURE 4 IMPURITY DISTRIBUTION • ARSENIC

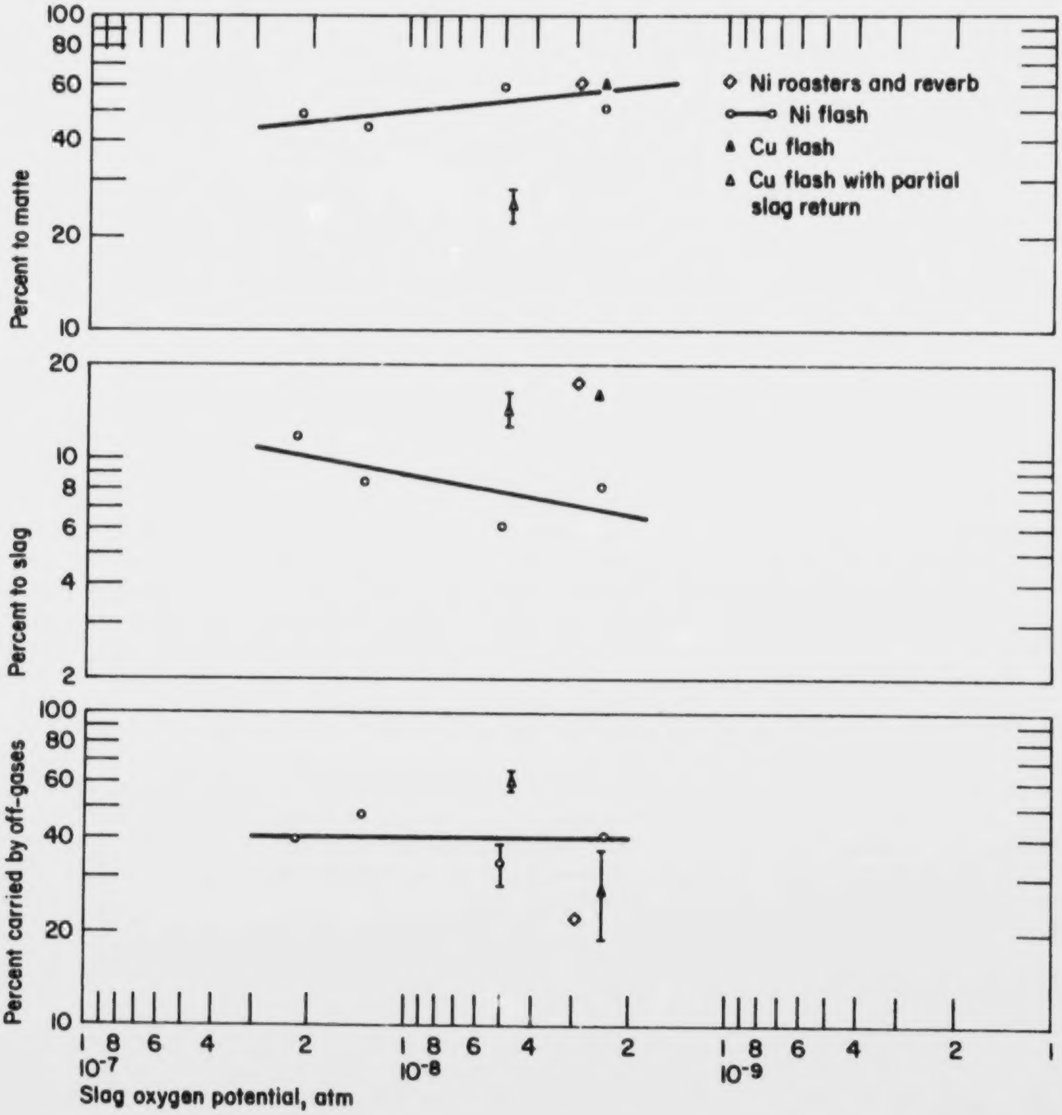


FIGURE 5 IMPURITY DISTRIBUTION • SELENIUM

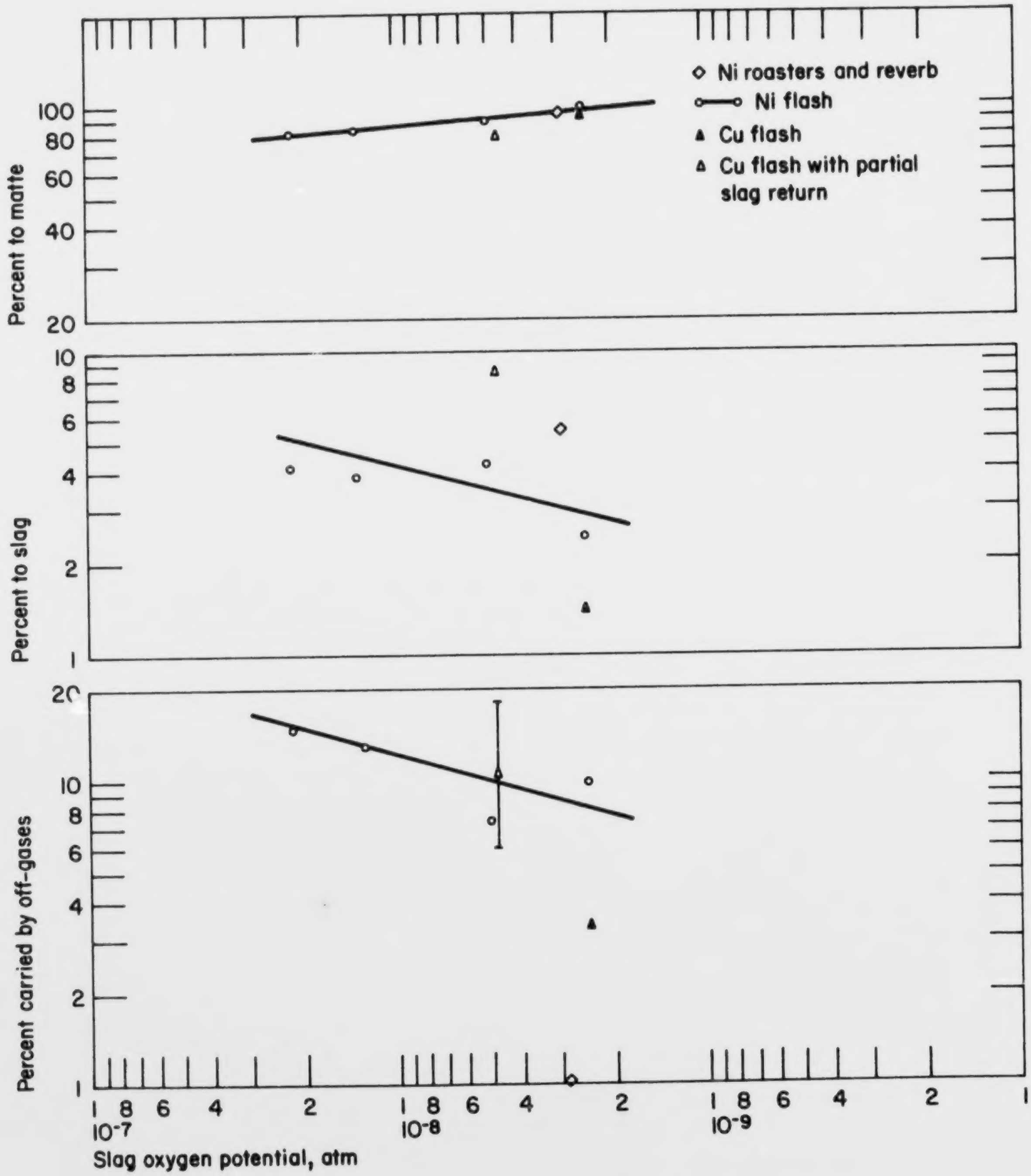


FIGURE 6 IMPURITY DISTRIBUTION • LEAD

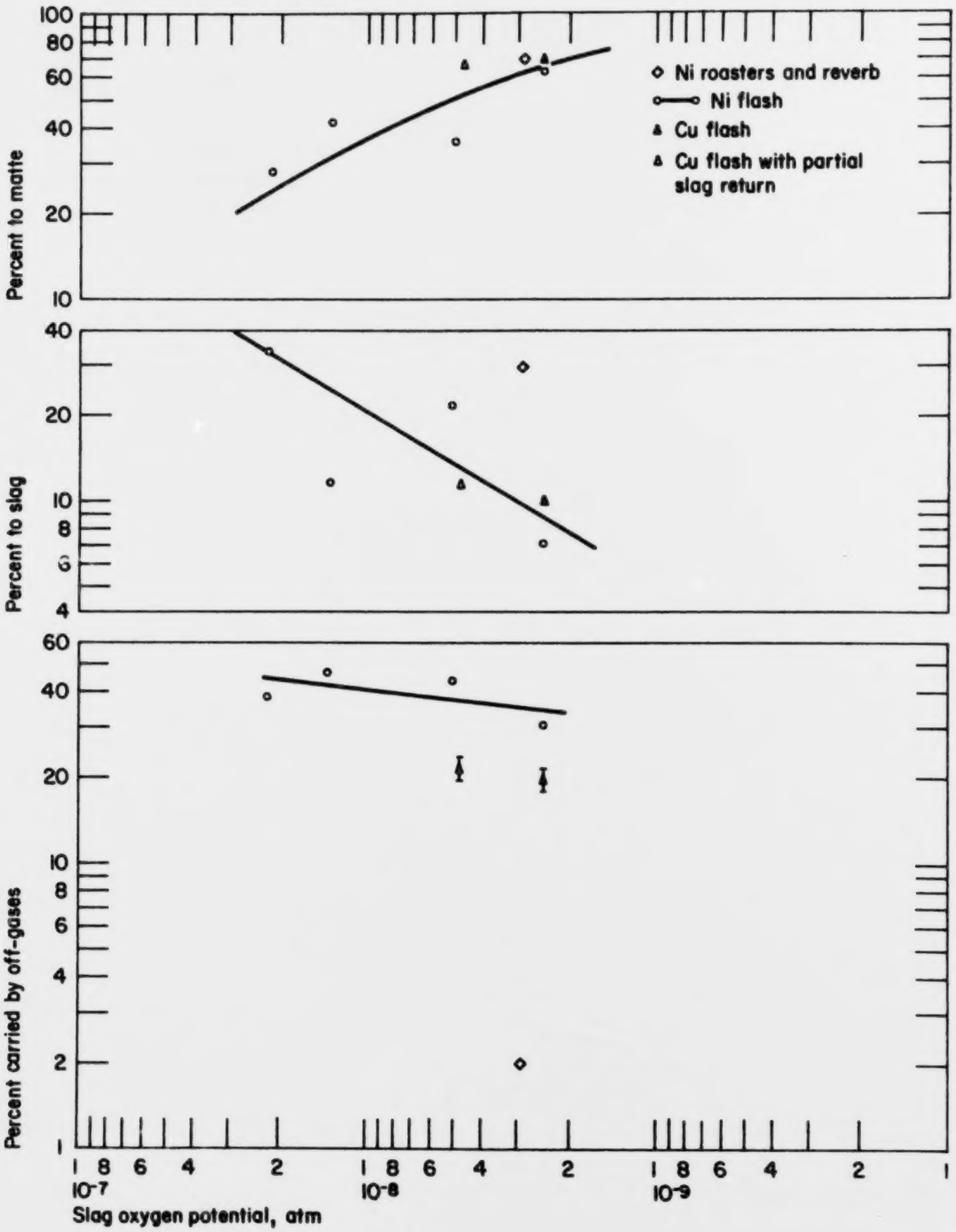
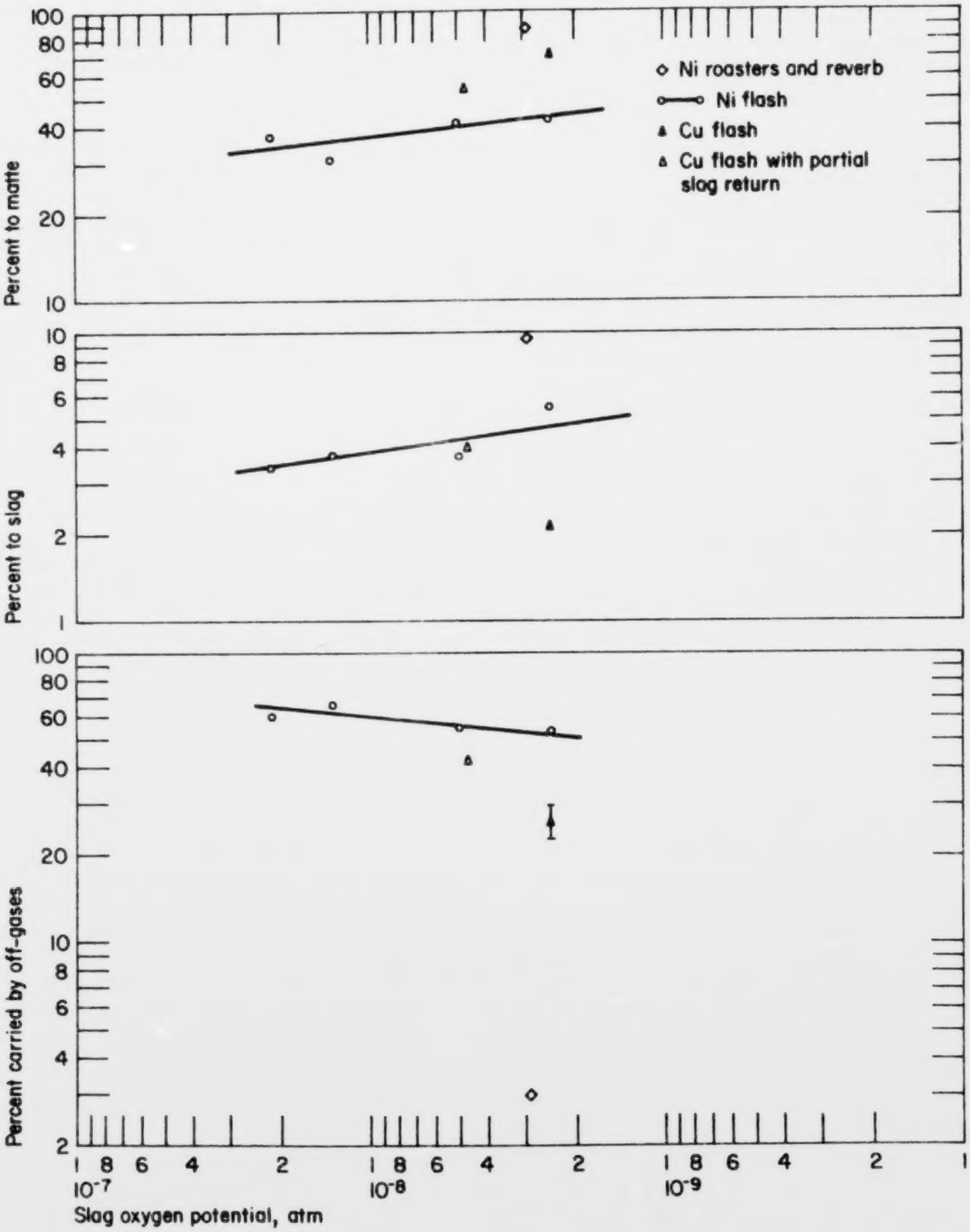


FIGURE 7 IMPURITY DISTRIBUTION • BISMUTH



that the process is generally regarded as one of the least (if not the least) expensive process in terms of energy consumption, capital expenditures and operating costs (17-19).

In the field of SO_2, particulate and impurity emission control, the oxygen flash smelting process offers further advantages which are best emphasized by focusing on the impact that its incorporation would have on a "typical", independent copper smelter.

Since sulfur dioxide and arsenic compounds are some of the species under closest scrutiny at this time in North America, sulfur and arsenic distributions were calculated, together with gas volumes and SO_2 concentrations, for the two flowsheets shown in Figures 8 and 9. Both plants were assumed to handle 1,100 tonnes per day of concentrate at 28% Cu, 29% Fe, 32.6% S and "low" arsenic, typical of many feeds being treated in the U.S. and Canada. To maintain a constant basis of comparison, the furnace matte grades were fixed at 42% in both cases.

The conventional copper smelter flowsheet involves:

- four multiple hearth roasters,
- one reverberatory furnace,
- three Peirce-Smith converters, one on stand-by and two operating at 50% blowing time,
- one 2160 m^3/min (81,000 scfm) acid plant handling converter gases at 4.4% SO_2 with 95% efficiency. Acid production would then be ∿570 tonnes/day.

By contrast, the oxygen flash smelter consists of:

- one 350 tpd oxygen plant,
- two fluid bed dryers,
- one oxygen flash furnace, without scrubbing system,
- the same converter operation, including slag return to the furnace,
- one 2275 m^3/min (85,000 scfm) acid plant handling the 7.4% SO_2 mixture obtained by enriching the converter gases with the flash furnace gases, 100 m^3/min at 75% SO_2. Acid production would then be 1010 tonnes/day. The acid plant sludge would either be discarded (at a loss of <0.5 percent of the copper in concentrate) or treated in a separate plant since it cannot be recycled to the smelter in this case because of impurity build-up considerations.

Sulfur and arsenic distributions in the various units were taken as:

FIGURE 8 SULFUR AND ARSENIC DISTRIBUTIONS

CONVENTIONAL REVERBERATORY SMELTING WITH SO_2 FIXATION ON CONVERTER GASES

Throughput = 1100 tonnes/day at 28% Cu, 29% Fe, 32.6% S, "low" arsenic

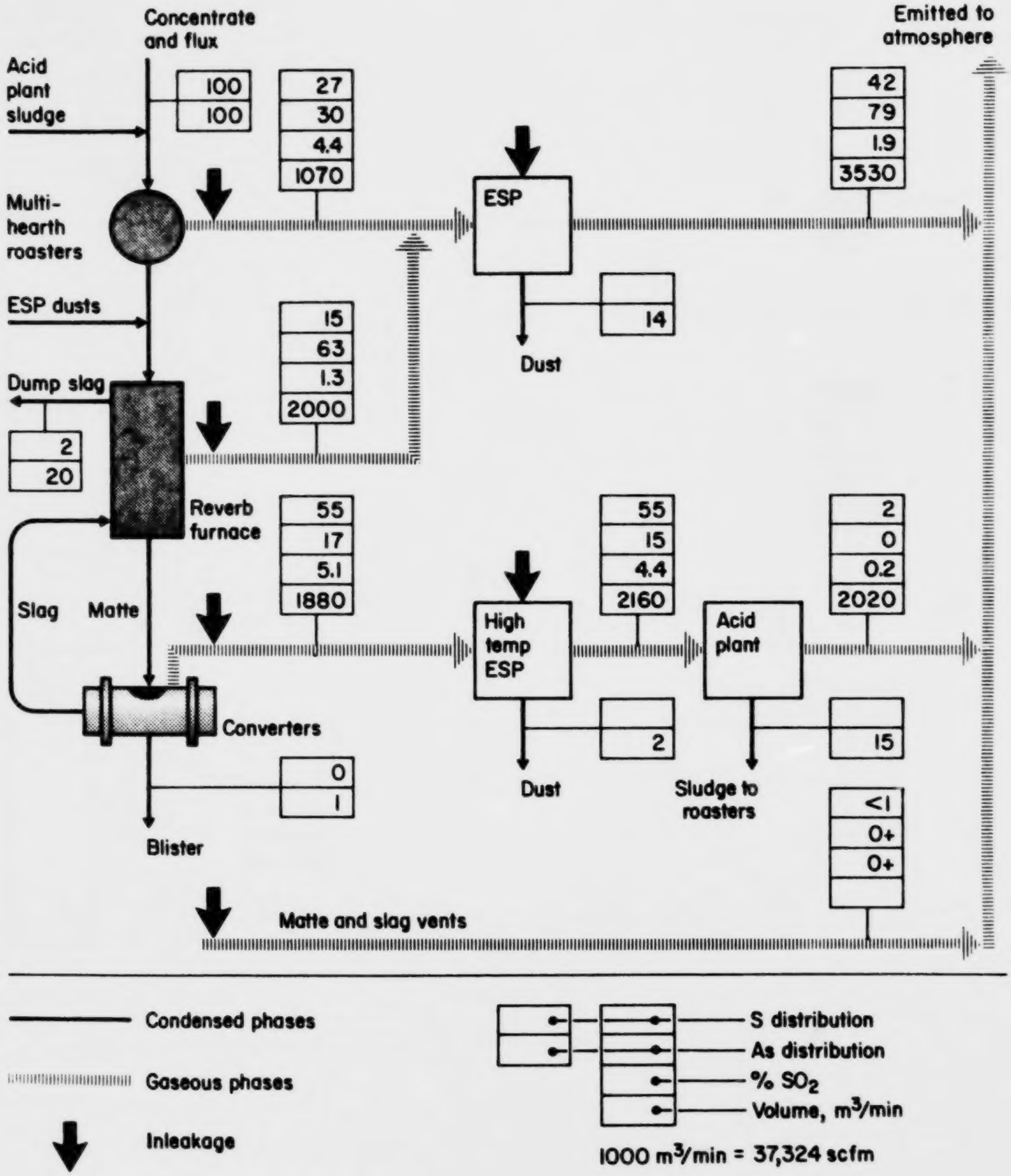
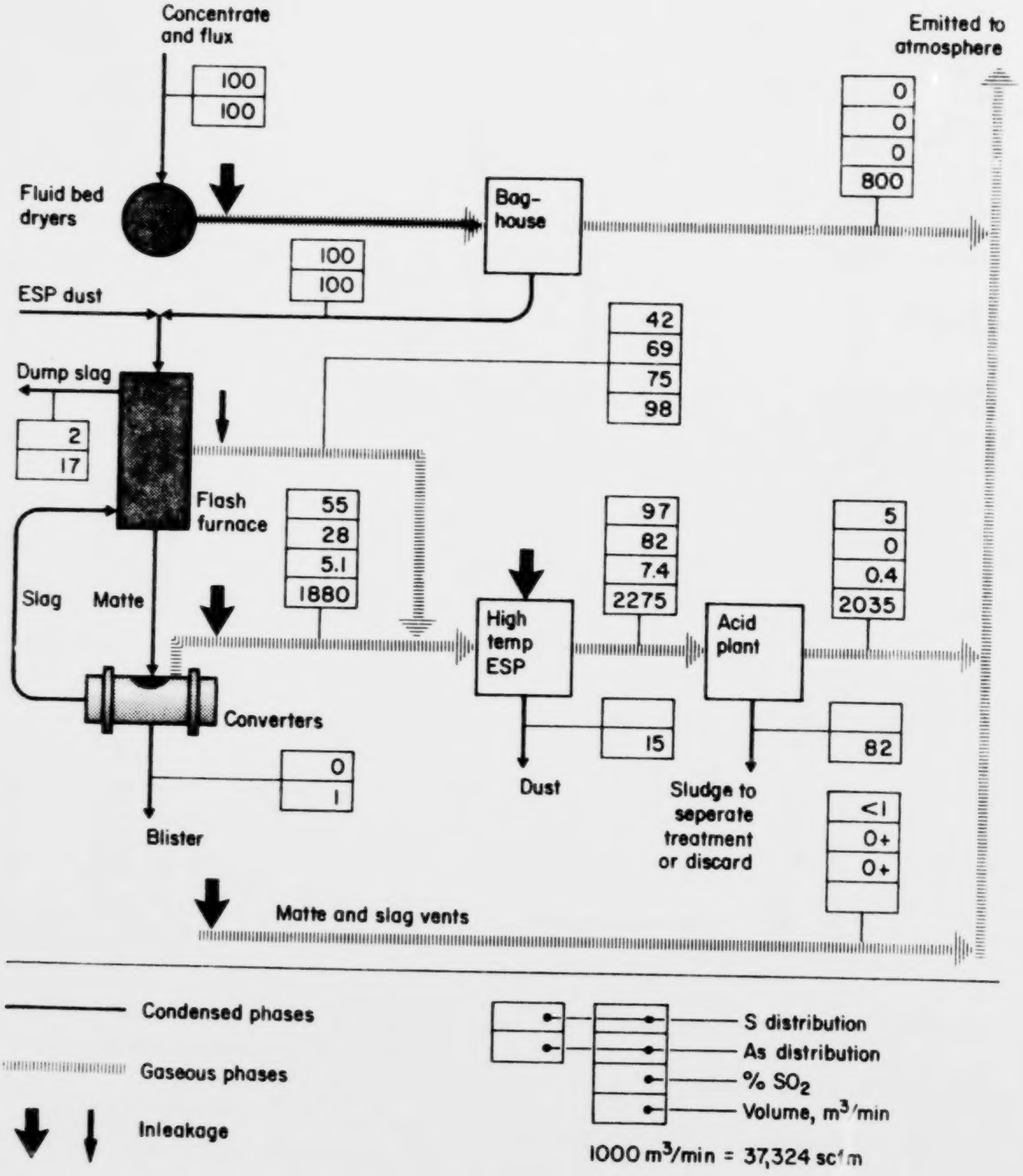

Concentrate and flux
100
100
Acid plant sludge
27
30
4.4
1070
Emitted to atmosphere
42
79
1.9
3530
Multi-hearth roasters
ESP
14
Dust
ESP dusts
15
63
1.3
2000
Dump slag
2
20
Reverb furnace
55
17
5.1
1880
55
15
4.4
2160
2
0
0.2
2020
Slag
Matte
High temp ESP
Acid plant
Converters
2
15
Dust
Sludge to roasters
0
1
Blister
<1
0+
0+
Matte and slag vents
Condensed phases
Gaseous phases
Inleakage
S distribution
As distribution
% SO_2
Volume, m³/min
1000 m³/min = 37,324 scfm

FIGURE 9 SULFUR AND ARSENIC DISTRIBUTIONS

OXYGEN FLASH SMELTING WITH SO_2 FIXATION ON SMELTING AND CONVERTING GASES

Throughput = 1100 tonnes/day at 28% Cu, 29% Fe, 32.6% S, "low" arsenic



Concentrate and flux
100
100
Emitted to atmosphere
0
0
0
800
Fluid bed dryers
Bag-house
100
100
ESP dust
42
69
75
98
Dump slag
2
17
Flash furnace
55
28
5.1
1880
97
82
7.4
2275
5
0
0.4
2035
Slag
Matte
High temp ESP
Acid plant
Converters
15
82
0
1
Dust
Blister
Sludge to seperate treatment or discard
<1
0+
0+
Matte and slag vents
Condensed phases
Gaseous phases
Inleakage
S distribution
As distribution
% SO_2
Volume, m^3/min
1000 m^3/min = 37,324 scfm

	Distribution, % of input to unit process			
	S		As	
	To Gas	To Slag	To Gas	To Slag
Multihearth roasters	27	-	27	-
Reverberatory furnace	20	3	62	20
Fluid bed dryers	0	-	0	-
Flash furnace	42	3	60	15
Converters	98	2	92	5

The diagrams clearly illustrate the potential impact that incorporation of the oxygen flash furnace would have on this "typical" copper operation. The two acid plants are virtually of the same size (∿2,200 m^3/min for the conventional smelter, 2,300 m^3/min for the flash smelter). However, because of the extremely small volume and high strength of the flash furnace gases blended with the converter gases, the flash smelter would have emissions of only ∿5% of the sulfur and virtually 0% of the arsenic. The conventional smelter, on the other hand, would emit ∿44% of its sulfur input and 79% of its arsenic input.

Reductions of the same magnitude would also be obtained on dust, lead, bismuth and other volatile impurities, since the low volume of the flash furnace gases allow their mixing with converter gases, normally the first stream to be controlled, without increasing this volume significantly. Incorporation of the oxygen flash furnace process in a conventional copper smelter with SO_2 fixation on converter gases could thus lead to virtually zero emissions without increasing scrubbing and acid plant facilities, the only other expense being disposal of the acid plant sludge.

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PROCESSING OF COPPER SMELTING GASES AT NAOSHIMA SMELTER

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ABSTRACT

The emission levels of sulfur oxides and particulate matter are strictly regulated in Japan. To comply with these regulations, all the gases produced from various furnaces, such as fluo-solid roasters, reverberatory furnaces, converters, and the new continuous smelting furnaces, are now treated through acid plants at the Naoshima Smelter of Mitsubishi Metal Corporation. At present, the Naoshima Smelter is the only copper smelting plant in the world which produces sulfuric acid from the calcine-fed reverberatory furnace gas low in sulfur dioxide concentration. This paper describes mainly the technical aspects of the total gas handling system at Naoshima with other features of the closed gas scrubbing solution treatment, gypsum production and newly installed desulfurizing plant for the acid plant tail gases.

INTRODUCTION

The Naoshima Smelter of Mitsubishi Metal Corporation is a custom smelter of copper located on the Naoshima Island in the Seto Inland Sea. The present total production capacity is 168,000 metric tons of electrolytic copper per year as shown below.

Production Capacity
(mt cathode/year)

Continuous smelter line	from concentrate 48,000
Reverberatory smelter line	from concentrate 96,000
	from scrap & blister 24,000
Total	168,000

The history of the Naoshima Smelter dates back to 1917 when the production started with a capacity of 3,600 metric ton blister/year by a green-charge reverberatory furnace line. The smelting capacity was gradually expanded thereafter together with various improvements and modifications. Then in 1969, an additional calcine-charge reverberatory furnace line was constructed to expand the over-all production capacity close to the present level.

On the other hand, it was becoming difficult to emit the reverberatory furnace off-gas to atmosphere under growing pressures from pollution-control requirements and the old No. 1 reverberatory smelter facilities were becoming obsolete. At the same time, the Mitsubishi Continuous Smelting and Converting Process was then under successful semi-commercial operation. Under such circumstances, the No. 1 reverberatory smelter was scrapped down in 1974 to be replaced by a new continuous smelter with a similar production capacity.

Concurrently with the replacement of the old smelter line, a series of pollution-control facilities were constructed, including an acid plant to treat low SO_2 gas from the reverberatory furnace, a plant to neutralize weak sulfuric acid bled off from the new acid plant with calcium carbonate for production of gypsum, and an effluent treatment plant to treat the washing water for the gas purification of the acid plants. For compliance with further stringent pollution-control regulations put into effect thereafter, the construction started for a desulfurization plant to treat the tail gas of the single-contact acid plant for the reverberatory furnace gas, which was completed and put into operation in March, 1978. The smelter operation has been stable with all the new facilities and the over-all sulfur fixation rate has now reached to 99%.

This paper outlines the present operation and describes the pollution-control measures incorporated at Naoshima.

OUTLINE OF SMELTER OPERATION

Raw Materials

During the initial period of operation, the Naoshima Smelter treated a considerable tonnage of domestic copper concentrate. At present, however, the domestic concentrate accounts only for 3% of the total concentrate treated due to a rapid increase of the production capacity and depletion of minable domestic ores. This percentage is expected to decrease further in future.

The Smelter is thus heavily dependent on raw materials imported from several foreign countries, such as Philippines, Australia, Canada, etc., which naturally differ from each other in tonnages and compositions. The Smelter is provided with a 50,000 mt capacity storage house to meet fluctuation in the tonnages of concentrates received, and with a bedding house to meet fluctuation in the composition and the contents of impurities. Secondary materials are also treated, mainly with converters, the tonnage of which tends to vary considerably depending on the prices of copper.

Smelting Step

The over-all flowsheet of the Smelter is shown in Figure 1. The major specifications of the reverberatory smelter facilities are given in Table 1.

In the reverberatory line, the bedded concentrate mixed with fluxes is first fed to a fluo-solid roaster for partial roasting, in which approximately 40% of the sulfur content is removed. The main portion of calcine (80 - 90%) is carried over with the gas from the roaster. The flux materials with under

Figure 1. Over-all flowsheet of Naoshima Smelter

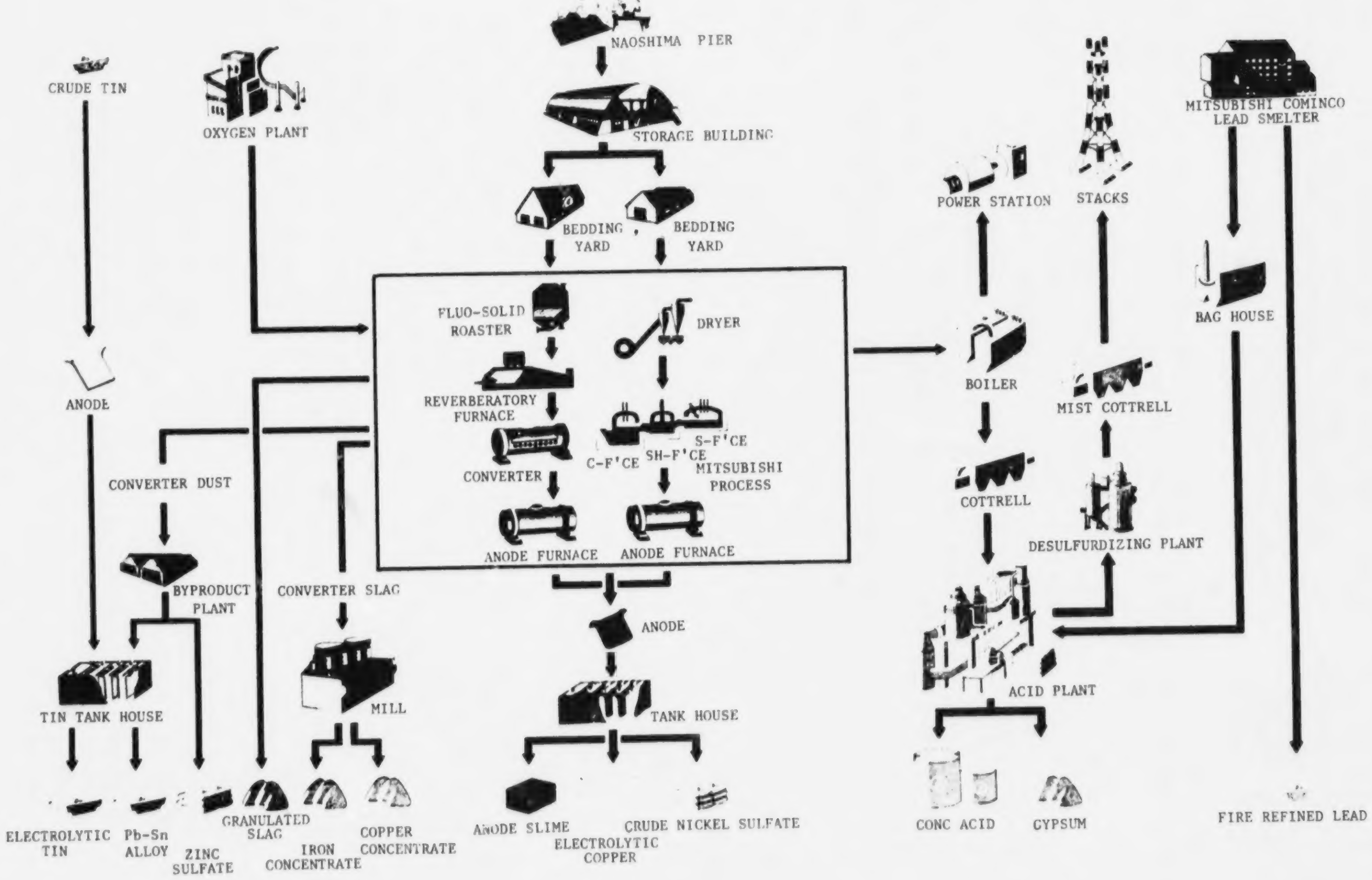

NAOSHIMA PIER
STORAGE BUILDING
BEDDING YARD
BEDDING YARD
CRUDE TIN
OXYGEN PLANT
ANODE
FLUO-SOLID ROASTER
REVERBERATORY FURNACE
CONVERTER
ANODE FURNACE
DRYER
C-F'CE
SH-F'CE
S-F'CE
MITSUBISHI PROCESS
ANODE FURNACE
CONVERTER DUST
BYPRODUCT PLANT
CONVERTER SLAG
MILL
TIN TANK HOUSE
ELECTROLYTIC TIN
Pb-Sn ALLOY
ZINC SULFATE
GRANULATED SLAG
IRON CONCENTRATE
COPPER CONCENTRATE
ANODE
TANK HOUSE
ANODE SLIME
ELECTROLYTIC COPPER
CRUDE NICKEL SULFATE
POWER STATION
BOILER
COTTRELL
ACID PLANT
CONC ACID
GYPSUM
STACKS
MIST COTTRELL
DESULFURDIZING PLANT
MITSUBISHI COMINCO LEAD SMELTER
BAG HOUSE
FIRE REFINED LEAD

Table 1. MAJOR SPECIFICATIONS OF SMELTER FACILITIES

Facility	Number	Specifications
Fluo-solid roaster	1	Dimensions: 5.3mφ x 8.0mL Charge : 55 wet ton/hr Bed temp : 630°C Boiler : 7.5 ton steam/hr
Reverberatory furnace	1	Dimensions: 9mW x 33mL Capacity : 1,200 ton charge/day Boiler : 33 ton steam/hr
Converter	3	Dimensions: 4m x 9mL Capacity : 150 ton blister/cycle Boiler : 15 ton steam/hr
Anode furnace	2	Dimensions: 4mφ x 10mL Capacity : 350 ton anode/batch

3 mm size, which act as a constituent to form a stable fluidized bed, are discharged through the roaster bottom. The off-gas is sent through cyclones for collection of calcine, cooled through a waste-heat boiler to approximately 350°C, passed through an electrostatic precipitator for removal of the dust content to 0.3 g/Nm^3 and then sent to an acid plant. Superheated steam recovered by the boiler is sent with the steam recovered by the reverberatory furnace boiler to a power generator. The calcine collected through the cyclones and the precipitator is transferred with the under flow materials by chain conveyor to the reverberatory furnace.

The calcine is fed to the reverberatory furnace by Wagstaff guns to be spread over the molten bath for quick smelting. Six low-pressure burners fire 5,500 ℓ/hr of heavy oil with the unit oil consumption of 110 ℓ/mt solid charge. Matte is controlled at 43% copper grade and discharged through four tapholes (two holes provided on each side of the furnace). Slag is skimmed through an overflow hole located near the furnace uptake, granulated by sea water and marketed. The off-gas is cooled through a waste-heat boiler from 1,300°C to around 350°C, passed through an electrostatic precipitator for dust removal to 0.1 - 0.3 g/Nm^3 and sent to an acid plant.

Three Pierce-Smith converters are installed, two in hot state and one on repair or cold standby. Each converter is equipped with 52 tuyeres, all punched by a Gaspe-type mechanical puncher. The off-gas is cooled through a boiler, passed through an electrostatic precipitator for dust removal to 0.3 g/Nm^3 and sent to an acid plant. Steam recovered by the converter boilers is used as process steam for the smelter and refinery operation, while the precipitator dust is sent to a dust treatment plant for recovery of valuable elements. The converter slag is slow-cooled and sent to a flotation plant to recover copper-rich concentrate and iron-rich concentrate, the former to be recycled to the fluo-solid roaster and the latter to be sold mainly to cement mills. By flotation of the converter slag, magnetite build-up in the reverberatory furnace was reduced and the over-all copper recovery was improved[1].

Two 350 mt capacity anode furnaces are provided, using ammonia as a reducing gas. Each anode weighs around 380 kg. The anodes are pressed by a straightening machine and sent to the tankhouse.

The Mitsubishi Process continuous smelter entered operation in March, 1974. The features of the Mitsubishi Process and the operation of the commercial unit have been described in the previous papers[2, 3, 4, 5, 6]. The off-gas from the smelting furnace and the converting furnace is cooled through the respective boiler, passed through an electrostatic precipitator and sent to an acid plant.

A lead smelter of Mitsubishi-Cominco Smelting Co., Ltd., a joint venture company between Mitsubishi Metal and Cominco Ltd., is located adjacent to the copper smelter. The lead smelter treats concentrate from the Pine Point mine with a production capacity of 36,000 mt pyrometallurgical lead per year. The off-gas from the sintering furnace is treated at an acid plant of the copper smelter.

Table 2 shows the production figures in May, 1978 at Naoshima.

POLLUTION CONTROL REGULATIONS

Along with rapid economical growth in Japan, there has been an increasing social pressure on control of pollutants generated at industrial plants since around the latter part of 1960's and the pollution-control regulations by the Central Government gradually became stringent. The emission level of SO_x is regulated by the so-called "k-value limitation" shown below.

$$q = k \times He^2 \times 10^{-3}$$

where q : Emission of SO_2 (Nm^3/hr)
He: Effective height of stack (m)
k: Constant

The emission level of SO_2 through each source is regulated by the above equation with the k value determined for the respective area. Since a higher stack would increase the allowable emission level, the industry constructed high stacks to comply with the regulations. Table 3 shows the change of the k value in the Noashima area together with the maximum ground level concentrations of SO_2 calculated in accordance with diffusion formula of Sutton and Bosanquet.

Since a few years ago, however, the local residents started to increase their voice for pollution control, particularly in the heavily-concentrated industrial areas, resulting in more stringent regulations contracted between

Table 2. TYPICAL PRODUCTION FIGURES
(May, 1978)

	(metric tons)
Continuous Smelter Line	
Concentrate smelted	16,100
Anode produced	4,500
Granulated slag	11,600
Reverberatory Smelter Line	
Concentrate smelted	28,200
Anode produced	12,800
(from concentrate)	(8,500)
(from blister & scrap)	(1,500)
(from anode scrap)	(2,800)
Granulated slag	10,600
Converter Slag Flotation	
Copper conc produced	1,100
Iron conc produced	8,000
Tankhouse	
Cathode produced	13,700
Crude Ni sulfate produced	20
Acid Plant	
Conc sulfuric acid	40,200
Neutralized gypsum	4,000
Desulfurdized gypsum	1,800

Table 3. CHANGE OF K VALUES IN NAOSHIMA AREA

Effective Date	K Value	Maximum Ground Level SO_2 Concentration (ppm)
June 1971	26.3	0.0447
April 1972	22.3	0.0379
June 1974	11.7	0.0199
December 1976	11.5	0.0197

the industries and the respective local government authorities. In most areas, these new regulations are based on the "total quantity emission limitation" and requires the control of the emission level of SO_2 regardless of the stack height.

The Naoshima Smelter entered into an environmental protection agreement with the local government with respect to both emissions and effluents, which came into effect in April, 1978. Table 4 shows the regulations by the local government for emission of SO_2, the SO_2 level in the environment, emission of particulate matters, and disposal of liquid effluents, as compared with those by the Central Government. For SO_2, these new regulations require that the total emission through the stacks of the Smelter should not exceed 295 Nm^3/hr and the average emission per day should not exceed 185 Nm^3/hr. For the ambient SO_2 concentration, the prefectural government can now order the Smelter to reduce the operation if the concentration of over 0.2 ppm continues for three hours, over 0.3 ppm for two hours, or over 0.5 ppm for one hour. All the emission levels are monitored by the telemetering system. At the Naoshima Smelter, the volume of, and the SO_2 strength in, the off-gas through the three major stacks, and the ambient SO_2 in the selected points on the Naoshima Island are constantly telemetered to the prefectural government. Under such regulations, even a minor misoperation would not be permissible.

While the emission levels of particulate matters and the disposal levels of liquid effluents are not yet telemetered, the regulations are far more stringent than those by the Central Government.

MEASURES FOR STRINGENT REGULATIONS

Gas Train System before 1974

Figure 2 shows the gas flow at Naoshima before 1974. The off-gas from the two reverberatory furnaces were, after dust removal, emitted to atmosphere. The roaster off-gas and part of the converter off-gas of the No. 1 reverberatory smelter was treated at the No. 1 Lurgi-type single-contact acid plant (called "L-1" at Naoshima) for production of concentrated sulfuric acid. The remaining part of the converter gas and the sintering furnace gas from the lead smelter were treated at the then-existing Petersen-type acid plant for production of 70% sulfuric acid. The tail gas from this acid plant was washed, together with the tail gas from the L-1, with sea water and emitted to atmosphere. The roaster gas and the converter gas of the No. 2 reverberatory smelter were treated at the No. 2 Lurgi-type double-contact acid plant (called "L-2" at Naoshima) for production of concentrated sulfuric acid. The tail gas from the L-2, with 200 ppm SO_2, was emitted directly to atmosphere.

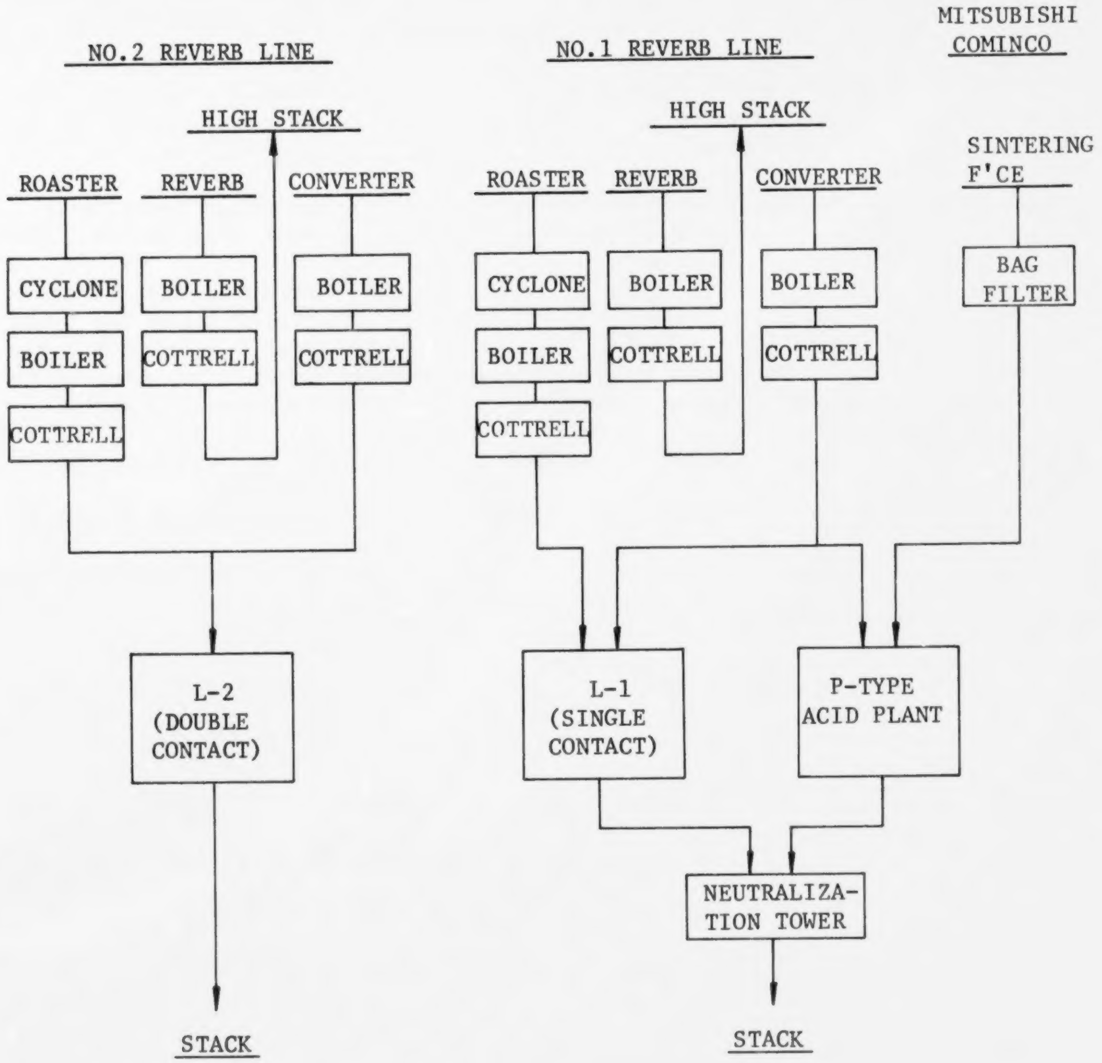
With the gas train system described above, the over-all sulfur fixation was 85%. It was then expected that direct emission of the reverberatory furnace gas to atmosphere would be restricted in future and there was concern over considerable leak gases at the old No. 1 reverberatory smelter.

Sulfuric Acid Production from Reverberatory Furnace Gas

Table 4. POLLUTION CONTROL REGULATIONS

	Central Government	Kagawa Prefecture
SO_2 Emission	Below k = 11.5 for each emission source	Maximum: below 295 Nm^3/hr Average per day: below 185 Nm^3/hr
Ambient SO_2	Average per day: below 0.04 ppm averaging the hourly values Maximum hourly value: below 0.10 ppm	20% operation reduction: over 0.2 ppm for 3 hrs over 0.3 ppm for 2 hrs over 0.5 ppm for 1 hrs 50% operation reduction: over 0.5 ppm for 2 hrs 80% operation reduction: over 0.5 ppm for 3 hrs over 0.7 ppm for 2 hrs
Particulate Matters	Total: below 200 mg/Nm^3 Pb : below 10 mg/Nm^3 Cd : below 1.0 mg/Nm^3	Total: below 100 mg/Nm^3 Pb : below 5 mg/Nm^3 Cd : below 0.5 mg/Nm^3 Cu : below 5 mg/Nm^3
Effluents	pH : 5 - 9 S.S.: 150 ppm COD : 120 ppm Zn : 5 ppm Cd : 0.1 ppm Pb : 1.0 ppm As : 0.5 ppm Cu : 1.0 ppm	pH : 5 - 9 S.S.: 35 ppm COD : 10 ppm Zn : - Cd : 0.05 ppm Pb : 0.5 ppm As : 0.3 ppm Cu : 1.0 ppm

Figure 2. Gas treatment flowsheet (before 1974)


NO.2 REVERB LINE
NO.1 REVERB LINE
MITSUBISHI COMINCO
HIGH STACK
HIGH STACK
SINTERING F'CE
ROASTER
REVERB
CONVERTER
ROASTER
REVERB
CONVERTER
CYCLONE
BOILER
BOILER
CYCLONE
BOILER
BOILER
BAG FILTER
BOILER
COTTRELL
COTTRELL
BOILER
COTTRELL
COTTRELL
COTTRELL
COTTRELL
L-2 (DOUBLE CONTACT)
L-1 (SINGLE CONTACT)
P-TYPE ACID PLANT
NEUTRALIZATION TOWER
STACK
STACK

Due to stringent pollution control requirements and economical reasons, it was decided in 1972 to replace the No. 1 reverberatory smelter with the new continuous smelter. The problem remaining to be solved was then treatment of the 1 - 2% SO_2 gas from the No. 2 reverberatory furnace.

Two alternative methods were studied for treatment of the reverberatory gas.

(1) Direct desulfurization of the gas to produce stable sulfur products, such as gypsum, or to produce concentrated SO_2 gas by an absorption-desorption cycle.

(2) Mixing the gas with the high SO_2 gas from the continuous smelter for production of sulfuric acid.

Various desulfurization methods were under development or in actual practice at that time, but all the processes had some disadvantages and no single process was available to meet all the requirements. The main problems of desulfurization were as follows.

(1) While less energy-consuming than acid production, desulfurization requires considerable costs for process supplies, such as a neutralizing reagent.

(2) There are few experiences with treatment of 1 - 2% SO_2 gas.

(3) If a solid material, such as lime, is to be used for neutralization, there is concern over various operational difficulties for handling of slurries throughout the plant.

On the other hand, the acid plant operation was fully established and reliable. Acid production had an additional advantage in that industrial-grade sulfuric acid was already produced in large tonnages and, among the sulfur compounds, most stable in prices and marketable tonnages. Reliability was given the highest priority in order not to disturb the smelter operation and it was finally decided to produce sulfuric acid from the reverberatory furnace gas.

While it would be possible to produce sulfuric acid only from the reverberatory gas, this process would consume considerable energy for water and heat balancing. The Onahama Smelter of Onahama Smelting & Refining Co., Ltd. experienced the production of sulfuric acid only from the 2.4% SO_2 gas from the green-charge reverberatory furnace[7]. This method was featured by the following.

(1) Use of a refrigerator in the gas purification step to reduce the gas temperature to below 6°C, thus maintaining the moisture content entering the converting step at a sufficiently low level.

(2) Constant firing of a preheater to provide supplemental heat.

The plant was, therefore, a large energy consumer and the operation was discontinued one year after the start-up at Onahama. The gas is now treated by a gypsum plant[8].

The off-gas from the calcine-charge reverberatory furnace at Naoshima is further lower in the SO_2 strength than at Onahama. If mixed with the gas from the continuous smelter, the SO_2 strength will increase to 4 - 5%, which is the minimum strength to self-sustain the heat balance in acid production without external heat supply.

At Naoshima, sea water is used as a coolant in the acid plants, the temperature of which rises close to 30°C in the summer season. With a carbon-cooler, the economical limitation of cooling is considered to be 40°C and as far as this system is employed, it would be impossible to maintain the water balance in summer.

Figure 3 shows water vapor pressures over sulfuric acid and the corresponding SO_2 concentrations required to maintain the water balance. As shown in the Figure, an increase of the acid concentration would decrease the water vapor pressure and the corresponding SO_2 strength. For example, if 50% sulfuric acid is to be circulated, the water balance can be maintained with the SO_2 strength of 2.8% even at the cooling temperature of 40°C. By application of this relationship, the new No. 3 acid plant (called "L-3" at Noashima) incorporated a drying tower using dilute sulfuric acid. Dilute sulfuric acid bled off from this drying tower is neutralized with limestone for production of gypsum, thus maintaining the water balance without a refrigerator. This method may be called "indirect desulfurization of the reverberatory furnace gas via sulfuric acid".

Other Measures

Concurrently with the construction of the L-3 and the neutralization plant, the following additional measures were taken.

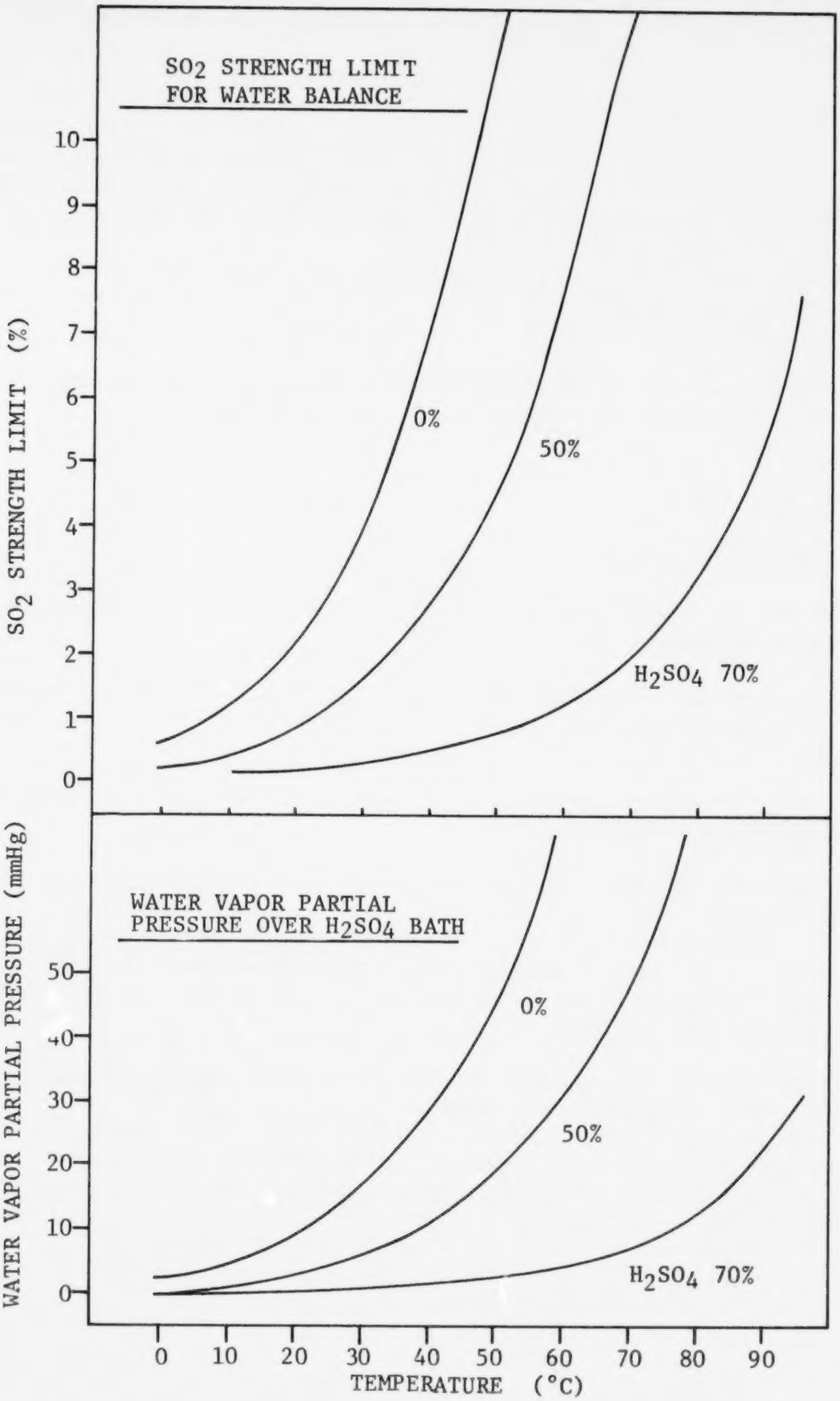
(1) The Petersen-type acid plant would be shut down due to obsolescence of the facilities and the high NO_x content in the tail gas.

(2) The tail gases from the three acid plants would be blended and passed through a mist precipitator to be newly installed and emitted to atmosphere through the stack 270 m high from the sea level.

(3) Scrubbing water in the gas purification step would be treated in the newly-installed liquid effluent treatment plant to cope with the expected regulations on the effluent control. While the waste scrubbing water contained volatile elements carried from the smelting operation, such as As, F, Pb, Zn and Se, it was only neutralized with NaOH before 1974.

(4) The spill gases around the furnaces, which were causing some problems to the ambient SO_2 concentration and the working conditions, would be collected as much as possible, passed through a bag filter for dust removal and then emitted to atmosphere through the high stack.

Figure 3. Vapor pressure and SO_2 strength over sulfuric acid



Measures for Pollution Control Agreement with Kagawa Prefecture

The following additional measures were taken for stringent pollution control requirements imposed by the prefectural government, which were all completed by March, 1978.

(1) The tail gas from the single-contact L-3 normally contained 2,000 ppm of SO_2, which corresponded to the SO_2 emission of 250 Nm^3/hr and already exceeded the limitation level. It was then decided to install a desulfurization plant. As to the process, a basic aluminum sulfate method was chosen for low investment cost, easy operation and low running cost.

(2) For the off-gases from heavy oil burners, which had been emitted directly to atmosphere, it was decided to use low-sulfur oil (less than 1.2% sulfur) instead of Bunker C oil containing 2.7% sulfur.

The current gas flow at Naoshima, incorporating all the measures described so far, is shown in Figure 4 together with the gas balance.

DESIGN AND CONSTRUCTION

L-3 and Neutralization Plant

Construction of the L-3 and the neutralization plant was started in 1973 and completed in February, 1974.

Figure 5 shows the flowsheet of the L-3 and Table 5 gives the major specifications as compared with the L-1 and L-2. The design features of the L-3 are as follows.

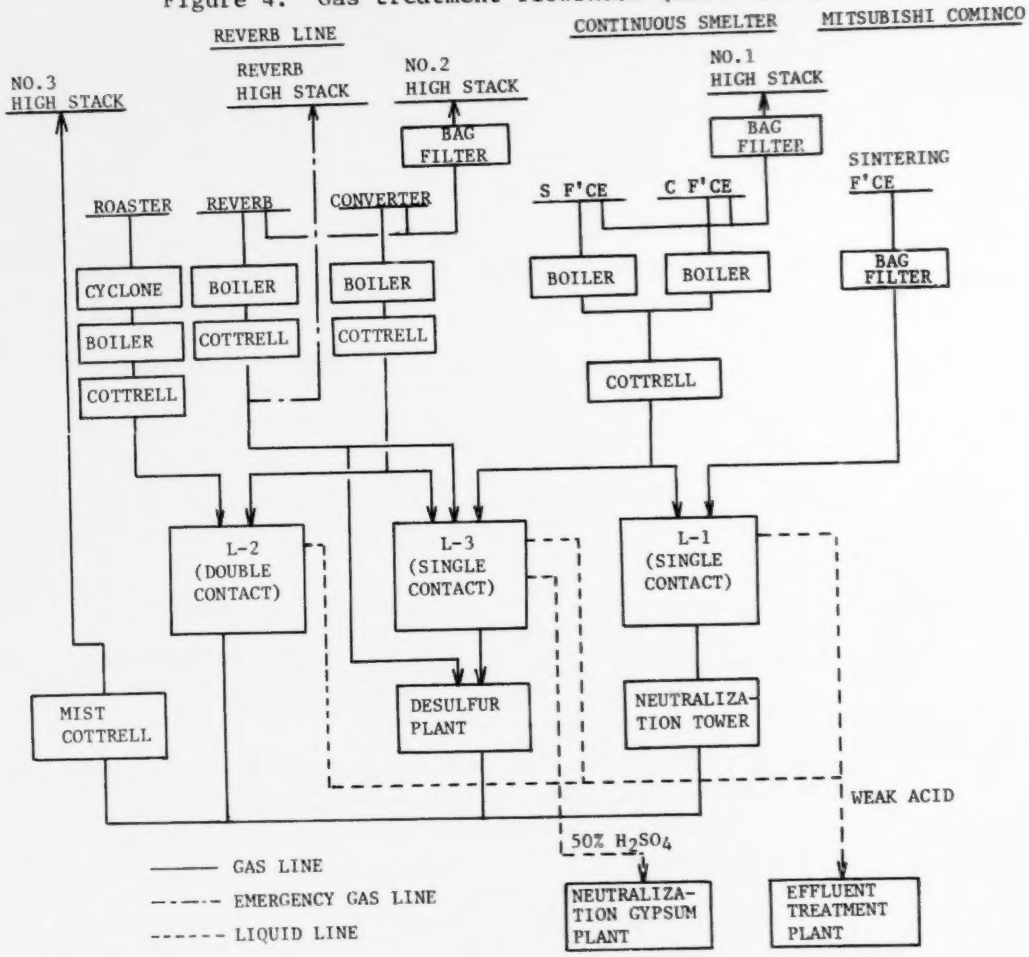
(1) The No. 1 drying tower using 45 - 60% sulfuric acid was installed to maintain the water balance.

(2) Heat-insulation of the converting section was strengthened to maintain the heat balance with the 4 - 5% SO_2 gas. In order to maintain as constant as possible the heat load in the converting section even with fluctuation of the gas volume and SO_2 strength, a fully automatic preheater was installed so as to make possible the operation only with the reverberatory furnace gas.

(3) The converters are designed so as to effectively absorb thermal expansions and shrinkages to prevent gas leakage.

(4) For cooling of concentrated sulfuric acid, an irrigation cooler was employed in the previous installations. However, the irrigation cooler had certain disadvantages, such as dissolution of Fe into acid, generation of steam mist and requirement of a large area for installation. Through the experiences at Onahama, it was also known that a considerable volume of NO_x would be contained in the acid produced from the reverberatory furnace gas, which would accelerate the corrosion of the cooler. Considering these factors, a Teflon-coil cooler was chosen for the L-3. There

Figure 4. Gas treatment flowsheet (after 1974)


REVERB LINE
CONTINUOUS SMELTER
MITSUBISHI COMINCO
NO.3 HIGH STACK
REVERB HIGH STACK
NO.2 HIGH STACK
NO.1 HIGH STACK
BAG FILTER
BAG FILTER
ROASTER
REVERB
CONVERTER
S F'CE
C F'CE
SINTERING F'CE
CYCLONE
BOILER
COTTRELL
BOILER
COTTRELL
BOILER
COTTRELL
BOILER
BOILER
COTTRELL
BAG FILTER
L-2 (DOUBLE CONTACT)
L-3 (SINGLE CONTACT)
L-1 (SINGLE CONTACT)
MIST COTTRELL
DESULFUR PLANT
NEUTRALIZATION TOWER
WEAK ACID
50% H2SO4
NEUTRALIZATION GYPSUM PLANT
EFFLUENT TREATMENT PLANT
GAS LINE
EMERGENCY GAS LINE
LIQUID LINE

GAS BALANCE

(Dry Nm^3/min)

	ROASTER	REVERB	CONVERTER	CONTINUOUS SMELTER	MITSUBISHI COMINCO	TOTAL
L-1				200	200	400
L-2	600		700			1,300
L-3		1,200	200	550		1,950
L-3 DESULFUR		200				200+α
TOTAL	600	1,400	900	750	200	

Figure 5. Flowsheet of L-3

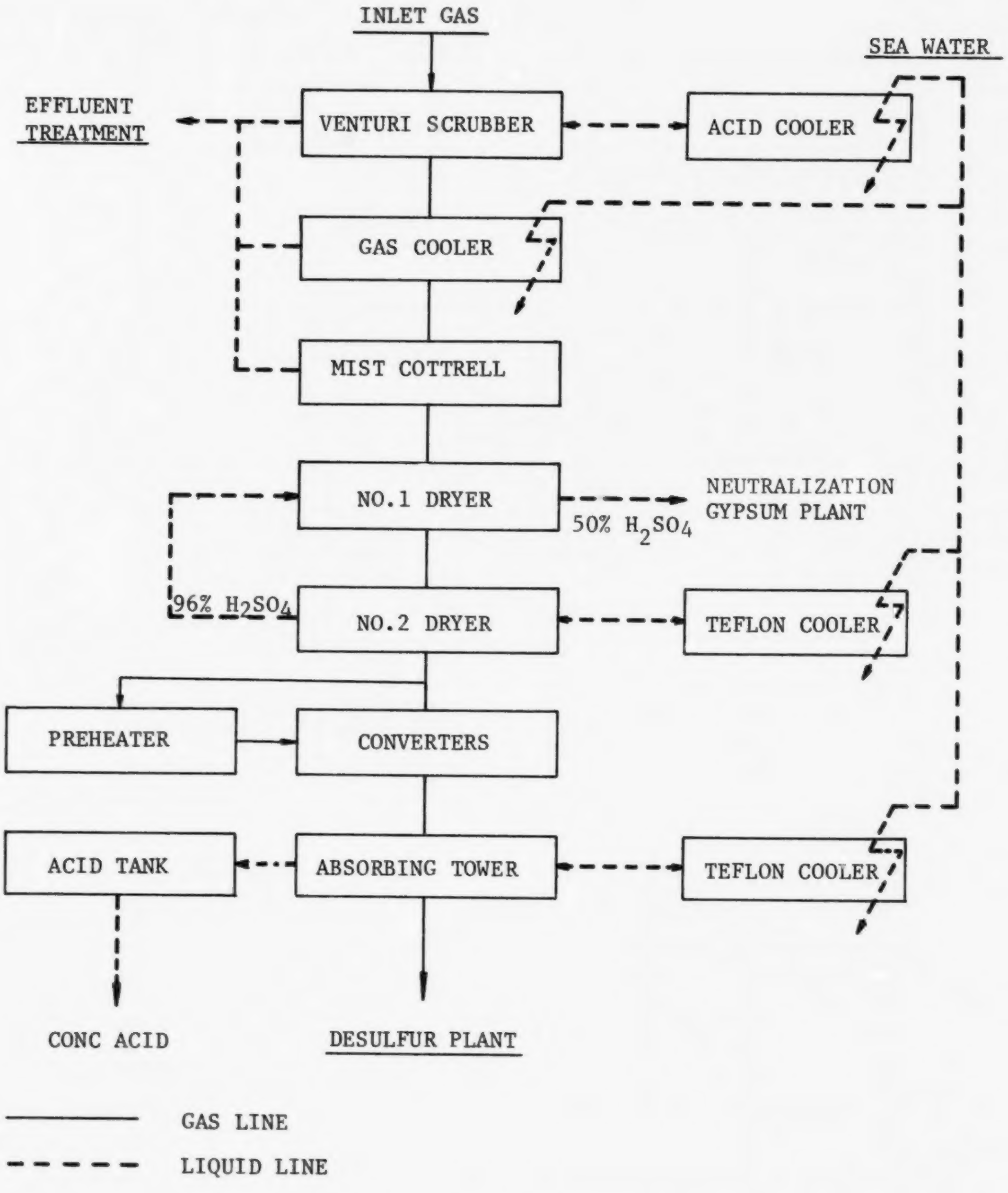

INLET GAS
SEA WATER
EFFLUENT TREATMENT
VENTURI SCRUBBER
ACID COOLER
GAS COOLER
MIST COTTRELL
NO.1 DRYER
NEUTRALIZATION GYPSUM PLANT
50% H_2SO_4
96% H_2SO_4
NO.2 DRYER
TEFLON COOLER
PREHEATER
CONVERTERS
ACID TANK
ABSORBING TOWER
TEFLON COOLER
CONC ACID
DESULFUR PLANT
GAS LINE
LIQUID LINE

Table 5. MAJOR SPECIFICATIONS OF ACID PLANT

	L-1	L-2	L-3
Type	Lurgi single contact	Lurgi double contact	Lurgi single contact
Capacity (t/d, 100% H_2SO_4)	240	700	500
Start of operation	1957	1969	1974
Main blower			
Number	2	2	2
Gas vol (m^3/min)	650	1,500	2,100
Pressure (mm Aq)	2,500	3,800	3,600
Power (KW)	380	2,200	2,500
Catalyst (m^3)	65	165	175
SO_2 in inlet gas (%)	av 6.0	av 8.5	av 4.5
Converting efficiency (%)	97	99.8	96
SO_2 in tail gas (ppm)	2,000	200	2,000

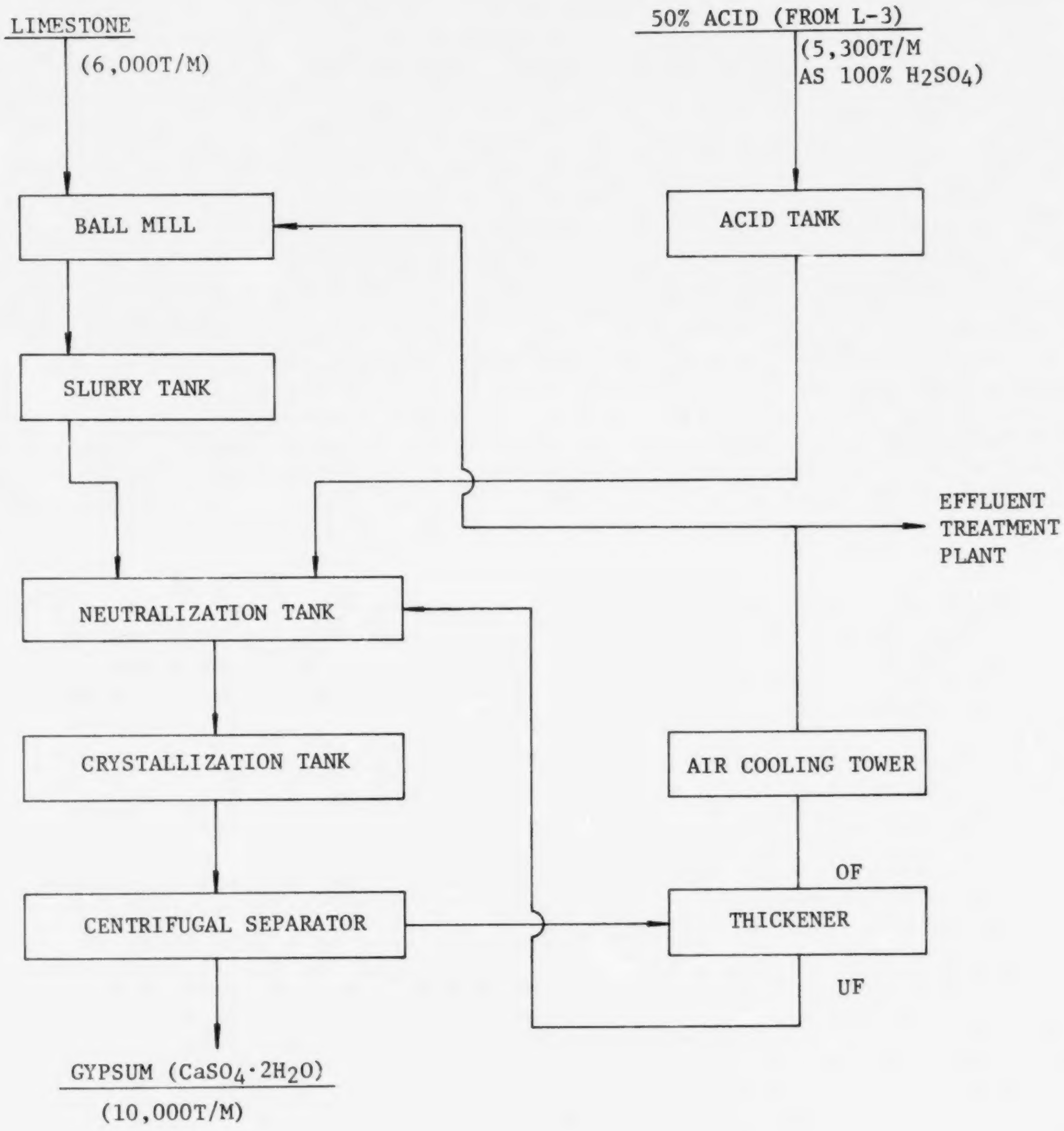
were, however, no actual experiences in Japan with the Teflon cooler in acid plants at that time and there was also some concern over plugging or damage of tubes by cooling sea water. After survey of the actual practices in foreign countries and extensive pilot plant tests at Naoshima, the Teflon cooler was finally employed.

(5) A tube-type mist precipitator was installed, thus improving the space velocity and reducing the installation area as compared with a conventional plate-type precipitator.

(6) FRP was used for the gas purification section. For the venturi scrubber, organic resin was used instead of glass fiber considering resistnace against fluorine.

Figure 6 shows the flowsheet and basic design figures of the neutralization plant. The plant has a capacity to produce 10,000 mt gypsum per month. The plant may be one of the largest plants in the world to produce gypsum directly from commercial-grade sulfuric acid. Prior to the plant design, a series of semi-commercial tests were conducted to determine the important parameters, such as pH, temperatures, residence time and recycled quantity of seed crystals. The maximum tonnage of dilute sulfuric acid which must be beld off from the L-3 to maintain the water balance in the summer season corresponds to the gypsum

Figure 6. Flowsheet of neutralization gypsum plant


LIMESTONE
(6,000T/M)
50% ACID (FROM L-3)
(5,300T/M
AS 100% H_2SO_4)
BALL MILL
ACID TANK
SLURRY TANK
EFFLUENT
TREATMENT
PLANT
NEUTRALIZATION TANK
CRYSTALLIZATION TANK
AIR COOLING TOWER
OF
CENTRIFUGAL SEPARATOR
THICKENER
UF
GYPSUM ($CaSO_4 \cdot 2H_2O$)
(10,000T/M)

production of 8,000 mt/month, and the tonnage is close to zero in winter. The plant was designed flexible to produce either sulfuric acid or gypsum within the allowable range. Gypsum produced is sent to an adjacent gypsum board plant by belt conveyor for manufacture of gypsum boards.

Liquid Effluent Treatment Plant

Figure 7 shows the flowsheet and normal operation figures of the liquid effluent treatment plant, which entered operation in January, 1974. The effluent from the acid plant purification section contains 2 - 3% sulfuric acid and considerable metal elements. Two-stage neutralization is, therefore, employed. The effluent is first mixed with calcium carbonate in the gypsum crystallization tank controlled in the pH range of 3 - 4, where the sulfate ion is separated as gypsum. The overflow from the No. 1 thickener is further neutralized with lime in the second neutralization tank controlled in the pH range of 10 - 11, where most metal contents are removed. For removal of As, $FeCl_3$ is added for coprecipitation with Fe. The sludge precipitated in the second neutralization tank is sent through the No. 2 thickener to be then filter-pressed. The overflow from the No. 2 thickener is neutralized with sulfuric acid to the pH of 7 and disposed. The tonnages of gypsum and sludge produced varies depending on the fluctuation of the tonnage of effluent treated. The average tonnages are 600 mt/month for gypsum and 150 dry mt/month for sludge.

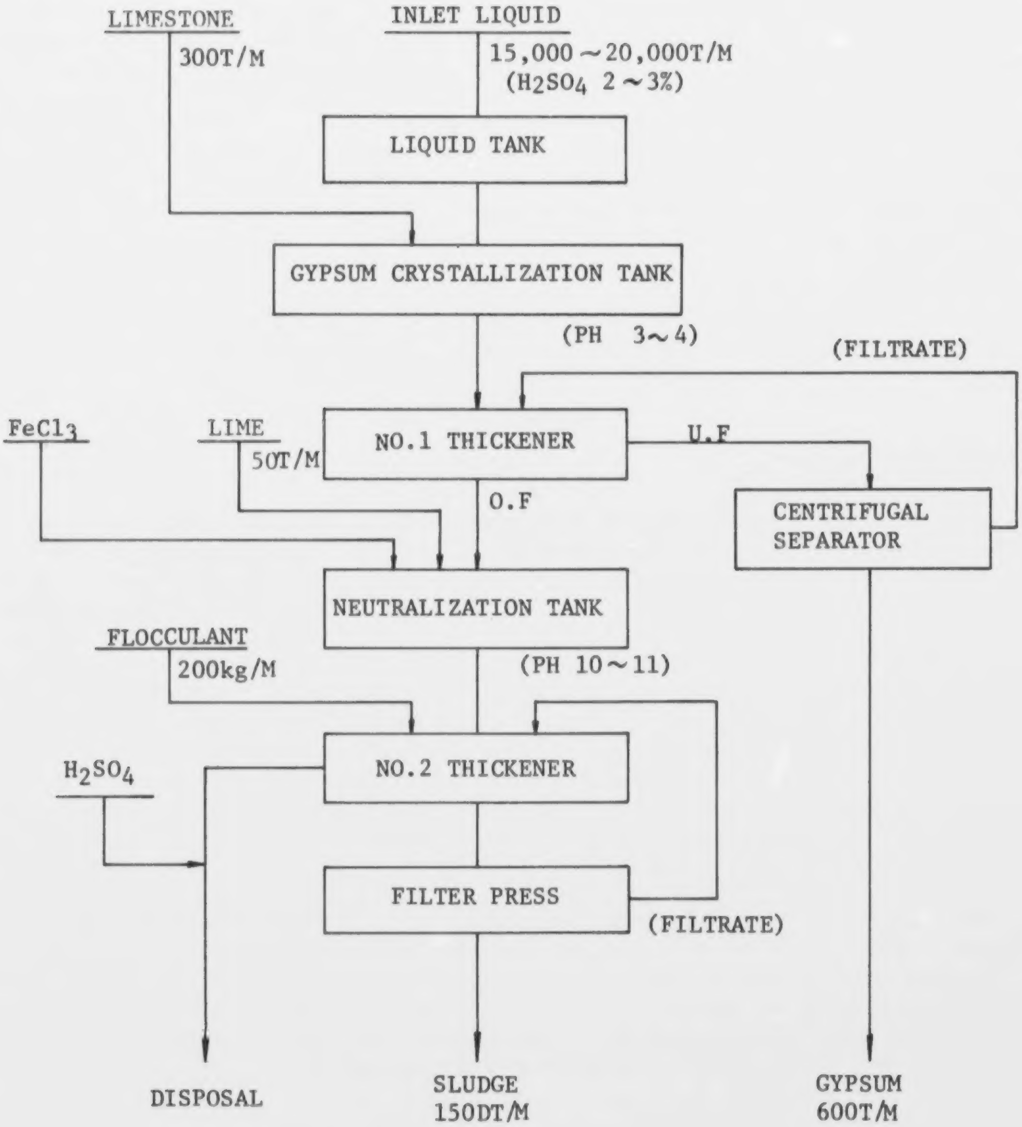
Spill Gas Treatment

For handling of spill gases, maximum possible enclosure with ventilation would be most effective and also desirable from the energy-saving standpoint. In conventional reverberatory smelters, a large volume of spill gas is generated for a short period of time during matte tapping, slag skimming, crane transportation of melts, and converter charging and discharging. Converters are particularly difficult to seal since the rope handling a ladle would come above the furnace. Control of the converter spill gases presents difficult problems in all the smelters in Japan. At Naoshima, the secondary hood was modified for higher efficiency and an air-curtain system was employed above the converter. The total volume of spill gases ventilated is now 4,500 Nm^3/min for both the reverberatory furnace and the converters, which is passed through a bag filter for dust removal and emitted through the No. 2 stack 270 m high from the sea level.

The continuous smelter employs stationary furnaces interconnected by launders. The volume of spill gases is, therefore, much smaller and, since the process is continuous, handling of the gases is much easier. Approximately 1,000 Nm^3/min of spill gases is ventilated, mixed with the off-gas from the concentrate dryer, passed through a bag filter for dust removal and then emitted through the No. 1 stack 170 m high from the sea level.

Acid Plant Tail Gas Desulfurization

Figure 7. Flowsheet of effluent treatment plant



LIMESTONE
300T/M
INLET LIQUID
15,000 ~ 20,000T/M
(H_2SO_4 2 ~ 3%)
LIQUID TANK
GYPSUM CRYSTALLIZATION TANK
(PH 3 ~ 4)
(FILTRATE)
$FeCl_3$
LIME
50T/M
NO.1 THICKENER
U.F
O.F
CENTRIFUGAL SEPARATOR
NEUTRALIZATION TANK
FLOCCULANT
200kg/M
(PH 10 ~ 11)
H_2SO_4
NO.2 THICKENER
FILTER PRESS
(FILTRATE)
DISPOSAL
SLUDGE
150DT/M
GYPSUM
600T/M

The acid plant tail gas desulfurization is based on the basic aluminum sulfate method, for which Dowa Mining Co., Ltd. owns the basic patent. The flowsheet is shown in Figure 8 and the main specifications in Table 6. The plant is designed to enable treatment of a part of the reverberatory furnace gas, in addition to the L-3 tail gas, so as to alleviate the operation load of the L-3.

Table 6. MAJOR SPECIFICATIONS OF DESULFURIZING PLANT

Type	Basic aluminum sulfate method
Inlet gas	L-3 tail gas 2,100Nm^3/min, 3,000ppm SO_2 Reverb off-gas 200Nm^3/min, 15,000ppm SO_2 Total 2,300Nm^3/min, 4,000ppm SO_2
Absorption efficiency	Over 95%
Gypsum produced	3,000 t/month (moisture 10%)
Blower	2,500Nm^3/min x 500mm H_2O 700KW
SO_2 in off-gas	Below 200ppm

For the acid plant tail gas desulfurization, a conventional gypsum method was initially considered. However, the basic aluminum sulfate process was found to have the following advantages.

(1) Liquid is handled through the absorption and oxidation section and there is no concern over plugging or erosion of the facilities associated with the slurry-handling system.

(2) Calcium carbonate is used as an absorbent instead of lime. In Japan, the price for one ton of calcium is 3 - 5 times more expensive with lime than with calcium carbonate.

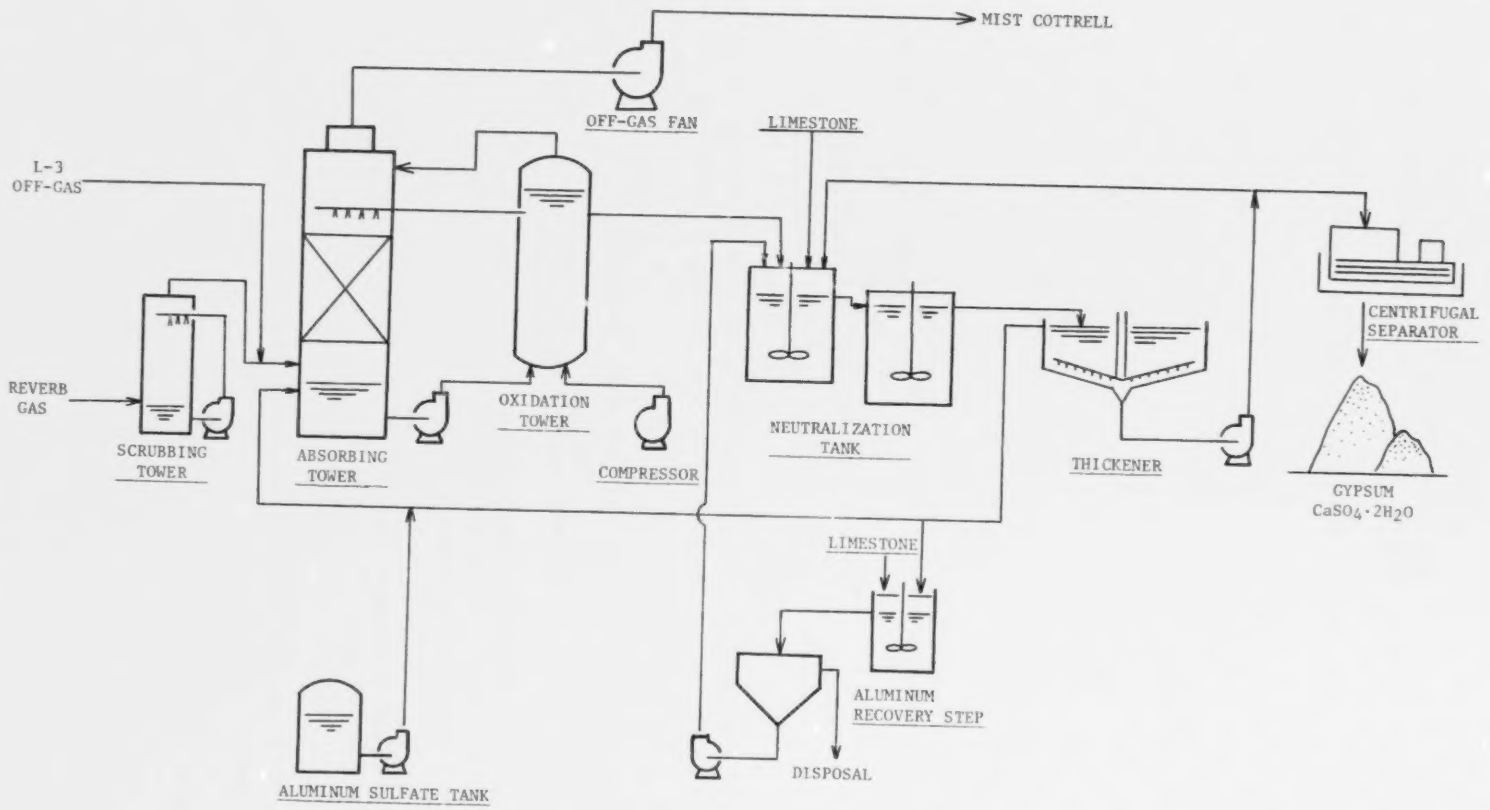
(3) The construction cost is lower and the over-all running cost, including energy consumption, is lower.

(4) The process is simpler to operate.

Reliability of the desulfurization plant is an extremely important factor in stabilization of the over-all smelter operation. For this reason, the basic aluminum sulfate method was finally chosen for the plant.

The outline of the process is as follows.

Figure 8. Flowsheet of desulfurization plant


MIST COTTRELL
OFF-GAS FAN
LIMESTONE
L-3
OFF-GAS
REVERB
GAS
SCRUBBING
TOWER
ABSORBING
TOWER
OXIDATION
TOWER
COMPRESSOR
NEUTRALIZATION
TANK
THICKENER
CENTRIFUGAL
SEPARATOR
GYPSUM
$CaSO_4 \cdot 2H_2O$
LIMESTONE
ALUMINUM
RECOVERY STEP
DISPOSAL
ALUMINUM SULFATE TANK

(1) Absorption step

In the absorption tower, the gas comes in contact with basic aluminum sulfate solution and desulfurized as follows.

$Al_2O_3 \cdot Al_2(SO_4)_3 + 3SO_2 \rightarrow Al_2(SO_3)_3 \cdot Al_2(SO_4)_3$ ....(1)

(2) Oxidation step

The solution is then oxidized with pressurized air in the liquid phase.

$Al_2(SO_3)_3 \cdot Al_2(SO_4)_3 + 3/2\ O_2 \rightarrow 2Al_2(SO_4)_3$ ....(2)

The liquid is handled up to this step without any slurry.

(3) Neutralization and filtration step

The oxidized solution is partly discharged to be neutralized with limestone for formation of gypsum.

$2Al_2(SO_4)_3 + 3CaCO_3 + 6H_2O \rightarrow Al_2O_3 \cdot Al_2(SO_4)_3 + 2CaSO_4 \cdot 2H_2O + 3CO_2$ ....(3)

The slurry after neutralization is thickened and fed to a centrifugal separator. The filtrate is recycled to the absorption tower. To prevent lowering of the aluminum concentration in the solution by water carried into the system, part of the filtrate is bled off to be sent to the aluminum recovery step and, after recovery of aluminum, the water is disposed.

(4) Aluminum recovery step

An excess quantity of calcium carbonate is added to precipitate aluminum hydroxide and dispose overflow water.

$Al_2O_3 \cdot Al_2(SO_4)_3\ (Aq.) + 3CaCO_3 + 8H_2O \rightarrow 3CaSO_4 \cdot 2H_2O + 4Al(OH)_3 + 3CO_2$ ....(4)

The precipitated aluminum hydroxide is recycled.

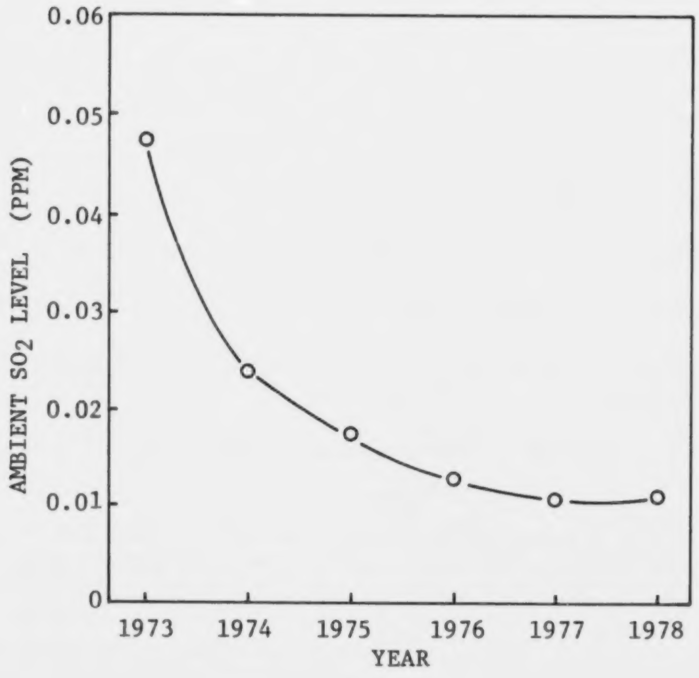
The by-product gypsum is transported by truck to the adjacent gypsum board plant.

ACTUAL OPERATION

After some start-up difficulties, the new gas treatment system is now continuing smooth operation.

Figure 9 shows the yearly averages of the SO_2 concentration at the location approximately 2 km from the Smelter. As shown in the Figure, the ambient SO_2 level has lowered year by year along with incorporation of various pollution

Figure 9. Change of ambient SO_2 concentration



control measures at the Smelter. The ambient SO_2 level is now nearly equal to that in the atmospheric background.

Table 7 shows the emission levels of SO_x and particulate matters. The present emission level of SO_x from the Smelter is considerably lower than the average limitation level of 185 Nm^3/hr imposed by the regulations. Table 8 gives a typical sulfur balance at Naoshima, which shows that the present sulfur fixation rate is 99%. As to the particulate matters, the present operation satisfies all the requirements. Generally, it is impossible to reduce the dust content in the furnace off-gas to less than 0.1 g/Nm^3 only with electrostatic precipitation. Since all the furnace gases are treated through acid plants at Naoshima, the emission of particulate matters through the No. 3 high stack is negligible as shown by the actual emission data. On the other hand, the gas emitted through the No. 1 high stack consists of the ventilated gas and the concentrate dryer off-gas, and the dust load is considerably high. The bag filters for dust removal are, therefore, under close vigilance and if the off-gas from the stack is colored even slightly, the bag filters are adjusted. The off-gas through the No. 2 high stack is only the ventilated

Table 7. EMISSION DATA

Gas		SO_2 Emission					Actual Emission Data of Particulate Matters (mg/Nm^3)			
		Regulated Levels (Nm^3/hr)		Actual Emission Data						
		Maximum	Normal	Gas Vol. (Nm^3/min)	SO_2 (ppm)	SO_2 (Nm^3/hr)	Total	Cu	Cd	As
No. 1 High Stack	Continuous smelter ventilation	44	25	1,500	200	18	80	4.2	0.3	2.5
No. 2 High Stack	Reverb smelter ventilation	90	70	4,300	194	50	30	2.5	0.1	0.5
No. 3 High Stack	Acid plants off-gas	140	80	3,900	180	42	5	Tr	Tr	Tr
Others		21	10			5				
Total		295	185			115				

Table 8. SULFUR BALANCE
(May, 1978)

	(ton)	(%)
Input		
Concentrate		
Reverberatory line	9,181	62.0
Continuous smelter line	4,993	33.7
Oil		
Reverberatory line	95	0.6
Continuous smelter line	32	0.2
Others	2	-
Mitsubishi-Cominco gas	514	3.5
Total	14,817	
Output		
Sulfuric acid*	13,369	90.2
Neutralized gypsum	674	4.5
Desulfurized gypsum	305	2.1
Effluent gypsum	100	0.7
Granulated slag (reverb)	80	0.5
Iron concentrate	9	0.1
Granulated slag (continuous)	70	0.5
Converter dust	20	0.1
Effluent disposed	60	0.4
Atmosphere	130	0.9
Total	14,817	

(Note) * The tonnage includes fuming sulfuric acid and liquid SO_2.

gas and presents no problems.

Smelter Operation and Gas Treatment

At Noashima, the gases from five sources are now treated by three acid plants. The gas flow interconnects the furnaces, acid plants and desulfurization plant by the respective booster fans and is considerably complicated. For this reason, the operation of each plant affects the over-all operation of the smelter. Reliability of each operation is, therefore, indispensable. If the operation of any one of the L-2, L-3 or desulfurization plant is suspended, the operation of the entire reverberatory furnace smelter must be also suspended. To cope with such possibilities, the blowers and other important devices are provided with stand-by units and the respective gas treatment plant is constantly under close vigilance. Various emergency measures are also incorporated. For example, inter-lock circuits are provided for the respective gas flow connections to prevent gas leakage from the furnace in case any blower of the gas treatment side fails.

Behavior of Impurities

The Naoshima Smelter treats more than ten different kinds of concentrates. Aside from precious metal elements, it is desirable that the concentrates are low in other impurities, such as As, Pb, Zn, Sb, F, etc., from the standpoints of pollution control, cathode quality and sulfuric acid quality. In the smelting step, such impurities are removed either with the discard slag or flue dusts.

At Naoshima, the converter precipitator dusts of the reverberatory smelter are all bled off and treated separately. The details of the dust treatment plant have been already reported[9]. At this plant, the respective impurities are fixed in marketable forms as follows.

Zn : zinc sulfate crystals
Cd : sludge
Pb and Sn: Pb-Sn alloy
Bi : Pb-Sn anode slime

In the continuous smelter line, the generation of mechanical dusts is smaller due to its unique feeding method. The dusts are in principle recycled to the furnaces, but if the impurities become concentrated in the gases, the precipitator dust is bled off[10]

Arsenic and fluorine are both highly volatile elements and also have high condensation temperatures. These elements are, therefore, difficult to collect in the precipitator. These impurities enter the acid plant, are fixed by scrubbing water and sent to the effluent treatment plant. Since arsenic is coprecipitated with iron in the effluent treatment plant, the arsenic content in the scrubbing water must be checked carefully. While fluorine is easily precipitated by neutralization, its corrosive properties cause problems in the gas purification section of the acid plant since some fluorine dissolves as HF and corrodes the refractories or FRP materials. Although the L-3 is provided with fluorine-resisting materials, corrosion of the refractories sometimes causes operational troubles in the L-1 and L-2.

The arsenic balance and the fluorine balance in case that the by-products of the effluent treatment plant are not recycled to the smelter are shown in Figures 10 and 11 respetively.

Previously, part of the gypsum was recycled to the smelter as a calcium source for slag making, while the sludge, with virtually no precious metal

Figure 10. As balance

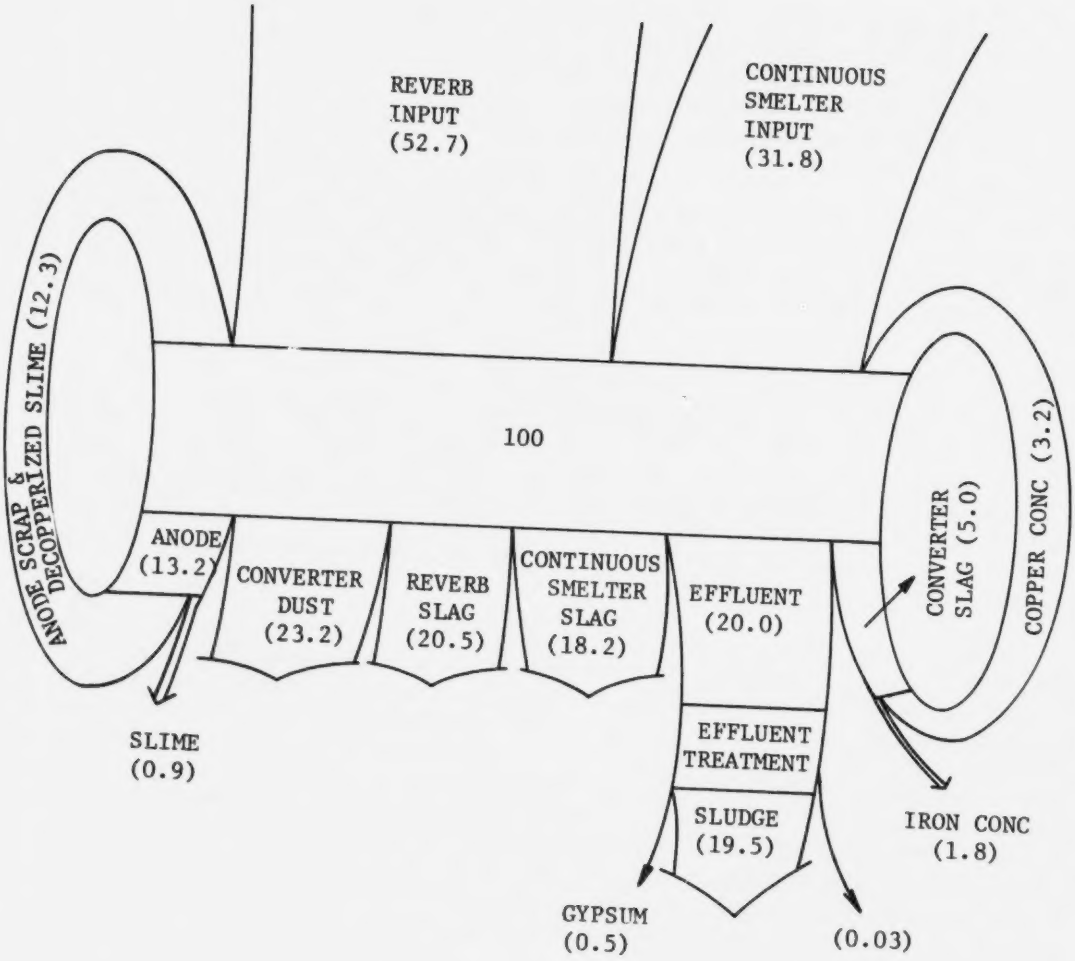
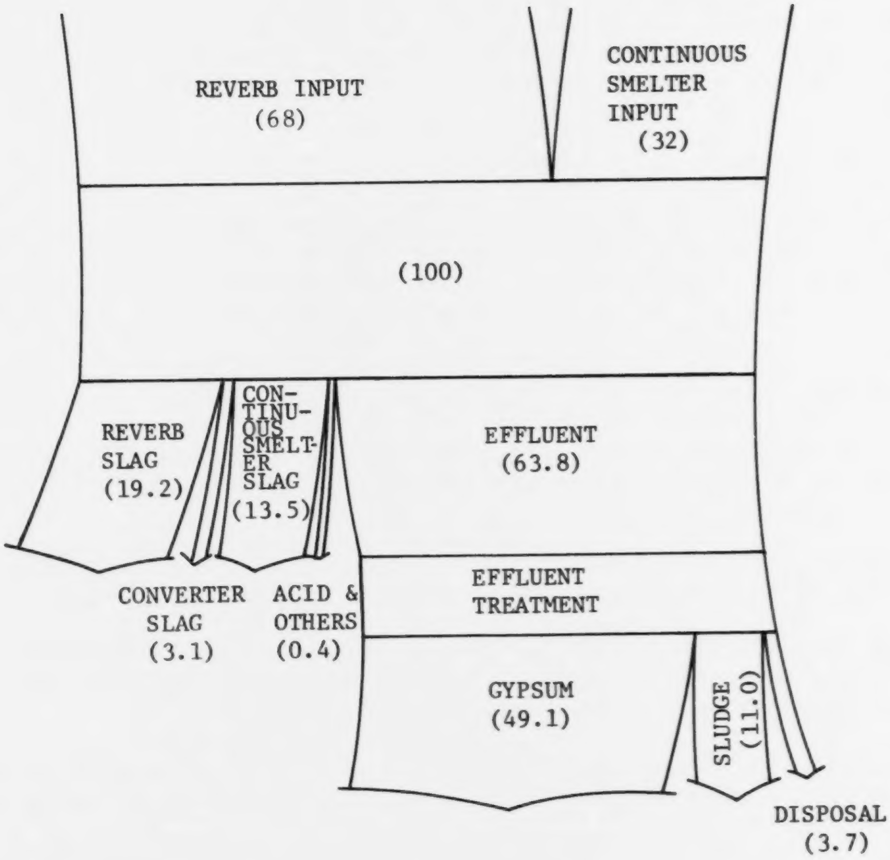

REVERB
INPUT
(52.7)
CONTINUOUS
SMELTER
INPUT
(31.8)
ANODE SCRAP &
DECOPPERIZED SLIME (12.3)
100
ANODE
(13.2)
CONVERTER
DUST
(23.2)
REVERB
SLAG
(20.5)
CONTINUOUS
SMELTER
SLAG
(18.2)
EFFLUENT
(20.0)
CONVERTER
SLAG (5.0)
COPPER CONC (3.2)
SLIME
(0.9)
EFFLUENT
TREATMENT
SLUDGE
(19.5)
GYPSUM
(0.5)
(0.03)
IRON CONC
(1.8)

Figure 11. Fluorine balance


REVERB INPUT
(68)
CONTINUOUS
SMELTER
INPUT
(32)
(100)
REVERB
SLAG
(19.2)
CON-
TINU-
OUS
SMELT-
ER
SLAG
(13.5)
EFFLUENT
(63.8)
CONVERTER
SLAG
(3.1)
ACID &
OTHERS
(0.4)
EFFLUENT
TREATMENT
GYPSUM
(49.1)
SLUDGE
(11.0)
DISPOSAL
(3.7)

contents and a high moisture content, was stored in the stockyard. Under pollution control regulations, however, simple storing of the sludge became difficult. There was also limitation on the space available for storage and it then became necessary to recycle the sludge to the smelter. However, if both gypsum and sludge are recycled, fluorine was expected to exceed the maximum allowance limitation. It was finally decided to market all the gypsum and recycle only the sludge. This recycling method started in March, 1978 and has been satisfactory to the present.

CONCLUSION

All the furnace gases are now treated through acid plants at Naoshima and the tail gas from the single-contact acid plant is desulfurized. The reverberatory furnace gas low in SO_2 is treated by indirect desulfurization combining an acid plant and a gypsum plant. The washing water for the acid plants is treated by the effluent treatment plant. The gas treatment plants are all in satisfactory operation, resulting in the over-all sulfur fixation of 99%.

ACKNOWLEDGEMENT

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PARTICULATE EMISSION CONTROL AT THE GARFIELD SMELTER

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ABSTRACT

The smelting system employed by Kennecott at the Utah Copper Division smelter at Garfield, Utah meets the particulate emission limitations which have been selected by EPA to achieve ambient air quality standards for particulate matter. This paper describes how this is accomplished. Furthermore, in achieving this clean air goal, we have identified an opportunity to reduce the cost of pollution cleanup by changing the manner in which emission limits are specified.

THE PROCESS

In the pyrometallurgical smelting of copper concentrates, particulate emissions and sulfur dioxide emissions share common sources of generation, namely the smelting vessels. The Utah Copper Division smelting system is composed of a two-stage process which is illustrated in Figure 1. The first stage is composed of three "reactor" smelting vessels which transform concentrate into an intermediate material called "matte" which contains approximately 70% copper. The production schedule utilizes two reactors in an operating mode with one down for maintenance. Matte is transferred from the reactors to the second stage of the smelting process via ladles carried by overhead cranes. The second stage of smelting is composed of four converter smelting vessels which transform the matte into blister copper.

PARTICULATE SOURCES

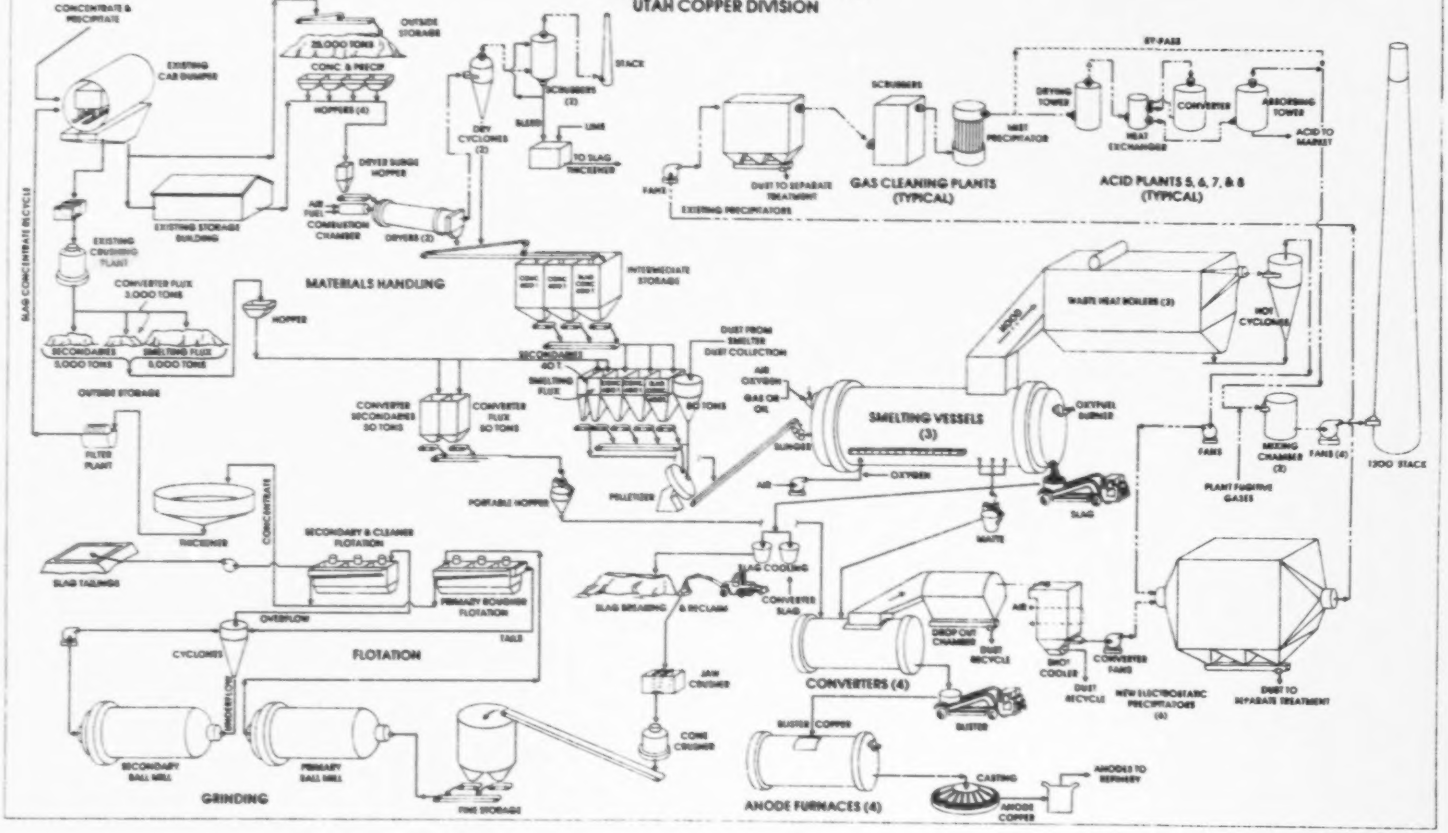
Particulates are generated in the smelting vessels in two different ways. First, condensed volatiles are particulates that form from metal fumes emitted from the molten furnace bath. As the gases from the vessels cool, the fumes condense into particulates. Second, the violent action of the process within the smelting vessels generates mechanically entrained dust which is carried away with the process gases. Another source of particulate is fugitive dust that is produced in the material handling systems and dust kicked up by the motion of heavy equipment and wind action in the plant area.

Figure 1. **FACILITIES FOR CONTROL OF SMELTER EMISSIONS**

PICTORIAL FLOW SHEET (SIMPLIFIED)

KENNECOTT COPPER CORPORATION

UTAH COPPER DIVISION


CONCENTRATE & PRECIPITATE
EXISTING CAR DUMPER
OUTSIDE STORAGE
25,000 TONS
CONC & PRECIP
HOPPERS (4)
DRYER SURGE HOPPER
AIR
FUEL
COMBUSTION CHAMBER
DRYERS (2)
DRY CYCLONES (2)
SCRUBBERS (2)
STACK
BLEED
LIME
TO SLAG THICKENER
EXISTING STORAGE BUILDING
SLAG CONCENTRATE RECYCLE
EXISTING CRUSHING PLANT
CONVERTER FLUX 3,000 TONS
SECONDARIES 5,000 TONS
SMELTING FLUX 5,000 TONS
OUTSIDE STORAGE
MATERIALS HANDLING
HOPPER
INTERMEDIATE STORAGE
SECONDARIES 40 T
SMELTING FLUX
80 TONS
CONVERTER SECONDARIES 50 TONS
CONVERTER FLUX 50 TONS
DUST FROM SMELTER DUST COLLECTION
AIR
OXYGEN
GAS OR OIL
SLINGER
PELLETIZER
PORTABLE HOPPER
FILTER PLANT
THICKENER
CONCENTRATE
SECONDARY & CLEANER FLOTATION
PRIMARY ROUGHER FLOTATION
SLAG TAILINGS
OVERFLOW
CYCLONES
TAILS
FLOTATION
UNDERFLOW
SECONDARY BALL MILL
PRIMARY BALL MILL
GRINDING
FINE STORAGE
FANS
EXISTING PRECIPITATORS
DUST TO SEPARATE TREATMENT
SCRUBBERS
WET PRECIPITATOR
GAS CLEANING PLANTS (TYPICAL)
BY-PASS
DRYING TOWER
HEAT EXCHANGER
CONVERTER
ABSORBING TOWER
ACID TO MARKET
ACID PLANTS 5, 6, 7, & 8 (TYPICAL)
HOOD
WASTE HEAT BOILERS (3)
HOT CYCLONES
SMELTING VESSELS (3)
OXYFUEL BURNER
OXYGEN
AIR
SLAG
MATTE
FANS
MIXING CHAMBER (2)
FANS (4)
1200 STACK
PLANT FUGITIVE GASES
SLAG COOLING
CONVERTER SLAG
SLAG BREAKING & RECLAIM
JAW CRUSHER
CONE CRUSHER
DROP OUT CHAMBER
DUST RECYCLE
AIR
SHOT COOLER
DUST RECYCLE
CONVERTER FANS
NEW ELECTROSTATIC PRECIPITATORS (4)
DUST TO SEPARATE TREATMENT
CONVERTERS (4)
BLISTER
BLISTER COPPER
ANODE FURNACES (4)
CASTING
ANODE COPPER
ANODES TO REFINERY

Two basic systems are used to control particulate emissions from the smelter. The primary gas handling system treats gases from both the reactors and converters. The secondary gas handling system collects the gases that escape from the primary sources. Gases from the primary gas handling system are cleaned in pollution control systems and then emitted from the 1200-foot stack. Gases from the secondary gas handling system are routed to the 1200-foot stack.

PARTICULATE CONTROL SYSTEM

By far the greatest amount of particulate is generated in the operation of the reactors and converters, so naturally the emphasis of the particulate control system is on the primary gas stream. Gases from the reactors and converters join at the new electrostatic precipitators (ESP's) and from there are treated in identical cleaning units. Before arriving at the new ESP's, however, the gases from the reactors and converters are treated somewhat differently. The gases from the reactors pass through water-cooled hoods to waste heat boilers where they are slowed down and the larger particulate drops out. The heat extracted from the gases is used to generate steam for plant use. From the waste heat boilers, the gases enter hot cyclones which serve to further remove particulate matter. Gases emerging from the hot cyclones pass through fans and are directed to the new ESP's.

The off-gases from the converters flow through water-cooled hoods to a dropout chamber where the larger particulate is removed. From there, they go to shot coolers where more particulate is removed before entering the new ESP's where they mix with the gases from the reactors.

The new ESP's serve to remove approximately 80% of the particulate. From the new ESP's the gases go to four separate acid plants or a visible train. Each of the acid plants is composed of a gas cleaning plant section and an acid production section. The gas cleaning section serves to remove the condensed volatile particulate before the gases enter the acid production section of the acid plant. The gas cleaning section is composed of three process steps. First, the gases pass through a humidifying tower where they are cooled and some particulate is removed. The second step is to a packed scrubbing tower where most of the condensed volatiles are removed. Finally, the gas is directed to a wet electrostatic precipitator where SO_3 aerosol mist is removed from the gas prior to entering the drying tower which is the first step of the SO_2 removal process. The visible train is simply the gas cleaning portion of an old acid plant. This was restored to serve as a backup whenever there is insufficient acid plant capacity to handle all of the gas in the SO_2 removal section of the acid plants. By doing this, we can remove the particulate from the process gas stream even when an acid plant is malfunctioning.

Another feature of the particulate control system is the secondary gas collection system which serves to control gases that escape from the primary sources. This is composed of secondary hooding around the reactors and converters and at the tapping ports on the reactors. These hoods remove gases that escape from around the primary hoods on the converters and reactors and also clean out the gases that escape during the slag and matte tapping operations at the reactor. The gases collected in the secondary gas collection system are directed to mixing chambers where they are blended with the outlet gases from the acid plants and then released from a 1200-foot stack.

SYSTEM PERFORMANCE

Most of the particulate in the stack gases comes from the secondary gas collection system. Consequently, the emission rate varies dramatically over the course of time. This is so because the operations such as tapping which produce much of the secondary particulate emissions occur intermittently. Another source of particulate variation, however, is the aerosol mist that is occasionally produced when an acid plant is malfunctioning or is experiencing a temperature unbalance. Taking these variations into account, the stack particulate emission rate can be expected to follow a statistical distribution such as the one shown in Figure 2. The average is 13.4 percent of the maximum, and the standard deviation is 17.6 percent of the maximum.

The foregoing result demonstrates an important fact that has been overlooked by most regulatory agencies in setting emission limitations for stationary sources. This fact is that constant emission control technology rarely, if ever, results in a constant rate of emission.

AN IMPROVED APPROACH TO EMISSION LIMITATION SPECIFICATIONS

Little, if any, attention has been given to how emission limitations should be specified and what effect such specifications have on the cost and effectiveness of environmental improvement efforts. This section analyzes the current practices for developing emission limit specifications and sets forth an alternative approach to specifying emission limits that in many cases can reduce control costs and eliminate litigation and controversy between regulatory agencies and sources.

Emission limitations have been based on two types of criteria. One approach has been to select emission limits directly on the basis of what is needed to meet ambient air quality standards. Another approach has been to base emission limits on the degree of control that can be achieved by specific technology--thus, the genesis of a series of four-letter symbols such as BACT, RACT and BCPT, etc., which denote various levels of control technology. In any case, the emission limitations that have been applied by regulatory agencies to most stationary sources do not reflect what is achievable or practical. This has resulted or can result in unnecessary expenditures in an attempt to achieve a degree of control that cannot be justified by any environmental benefits.

The current practice for specifying emission limits on the basis of ambient standards is to use the "worst case" approach. This involves a three-step procedure. First, a maximum ambient concentration is estimated by use of a diffusion model or it is obtained from actual field measurements. Next, an emission which produced the maximum concentration is

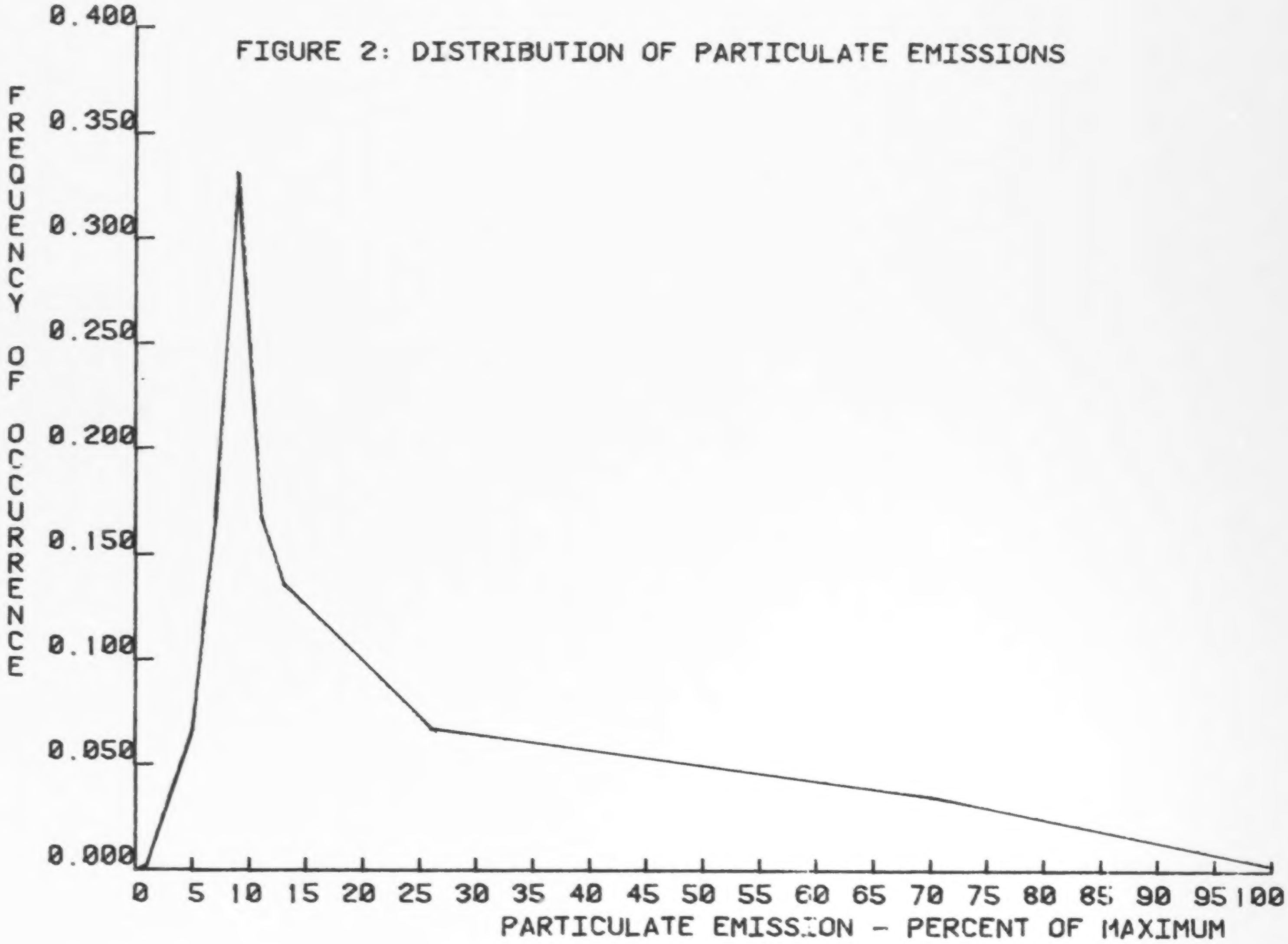


FIGURE 2: DISTRIBUTION OF PARTICULATE EMISSIONS

estimated or obtained from measurements, and finally, a proportional or rollback calculation is performed to determine the maximum emission that would be permitted without exceeding the worst case concentration. The result of this procedure is the specification of a single fixed maximum emission rate for the source. This specification is often based on two uncertain parameters. First, the maximum concentration can rarely be determined with a degree of precision that is anywhere near what is needed for the development of a fixed result, and second, the emission rate corresponding to the maximum concentration is rarely known if for no other reason than the transport delay between the source and receptor precludes a precise correlation between concentration and emission rate. Consequently, this procedure utilizes variables that are subject to a great deal of uncertainty and pops up with a resultant number that must be achieved with absolute certainty. Now this is inconsistent with the way in which ambient standards are specified. The ambient standards are set on a statistical basis, that is, a given concentration cannot be exceeded more than a certain percent of time. Most stationary sources do not emit at a constant rate; therefore, in order to determine whether the emissions from the sources will exceed the ambient standards, one must evaluate the probability that the emission rate will be high enough to exceed the standard at the same time that worst case dispersion conditions exist. If the probability that the emission rate will exceed a certain value at the same time that the poorest atmospheric dispersion conditions exist is sufficiently small, then ambient standards will not be exceeded. This suggests the notion of a statistical specification for emission limitations instead of the current practice of specifying a single maximum emission limit.

When emission limitations are based on control technology, a similar problem arises. The present practice is to estimate the efficiency of various pieces of control equipment operating under ideal conditions and then calculate an emission limit based on a fixed or constant input to the control equipment with a constant resultant output. Most processes--and this is particularly true of smelting processes--do not produce constant outputs of either product or gas streams. Furthermore, most control equipment does not operate at a constant efficiency. Combining these two variable factors results in an emission output that is variable, not one that is constant. Again, the variation in the output depends on the variations in the process and in the control equipment. These variations in output from the control equipment result in an emission that has statistical properties. Generally, regardless of the nature of the process or control device, the resulting emission limitation would be expressed or specified in terms of a statistical distribution. By doing this, the emission limitation can be made to reflect a realistic assessment of what technology can actually achieve.

If emission limitations were specified as a statistical distribution, consistent with the performance of the applicable control technology, substantial cost savings could be achieved. First, attempting to add controls such that the maximum emission is equivalent to the average emission achievable by a given technology can be extremely costly. This is because such controls require backup equipment or production curtailments, either of which is costly. Second, even though some regulations have permitted excursions over a fixed limitation under certain malfunction conditions, the cost of analyzing and filing the reports to explain the excursion can be substantial for both the source and the regulatory agency. A statistical specification of emission could, in many cases, eliminate these costs and make the achievement of environmental goals more economical.

THE RECOVERY OF PARTICULATE MATTER AND SULFUR DIOXIDE AT THE HIDALGO SMELTER

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Abstract

The Hidalgo Smelter of Phelps Dodge Corporation in New Mexico utilizes the unique flash smelting process developed in Finland to achieve primary smelting of copper. The flash smelting process generates a high-strength sulfur dioxide gas stream that may be treated by various processes to allow the removal of the sulfur dioxide gas. At the Hidalgo Smelter the continuous gas stream of high-strength sulfur dioxide from the flash furnace is mixed with the intermittent, relatively low-strength sulfur dioxide gas stream from the converters prior to treatment at the sulfuric acid facilities. The continuous feed stock of relatively high-strength sulfur dioxide gas enables the sulfuric acid plants to maintain a stable and efficient operation.

High efficiency electrostatic precipitators are installed at the Hidalgo Smelter to remove particulate matter from the flash furnace off-gases as well as from the rotary dryer off-gases. The efficient operations of the electrostatic precipitators and the sulfuric acid plants have enabled the smelter to comply with environmental regulations governing particulate emissions and sulfur capture.

General Description of the Hidalgo Smelter

The Hidalgo Smelter consists of a rotary dryer, a flash smelting furnace, an electric furnace, three converters, two anode furnaces with a casting wheel, two sulfuric acid plants and an elemental sulfur plant. The elemental sulfur plant is not presently in operation. Supporting facilities include a power plant, a crushing plant, a bedding plant, an analytical laboratory and various shops and warehouses.

The copper concentrates and silica flux are received by rail. The concentrates are blended in the bedding plant and reclaimed by a bucket-wheel reclaimer for transport to a rotary dryer. The concentrates and silica flux, which contain approximately six to eight percent moisture, are dried in the dryer to approximately 0.2 percent moisture prior to transportation to the furnace feed system by a pneumatic lift. Exhaust gases from the dryer pass through a high efficiency electrostatic precipitator to remove the particulate matter prior to discharge to the atmosphere through the 600-foot stack.

The dried mixture is fed by drag conveyors to the flash furnace through four concentrate burners where the concentrates react with the oxygen in the preheated process air. Preheated air and fuel oil are controlled by computer depending on the chemical composition of the concentrates and the desired matte grade. The furnace gases containing about ten percent SO_2 travel through the settler portion and the uptake shaft of the flash furnace prior to entering the waste heat boiler. The furnace gases are cooled from about 2,200° F to 650° F in the waste heat boiler to produce saturated steam which is superheated for use in power generation. The cooled gases enter three electrostatic precipitators in parallel where the majority of the particulate matter is removed. Three induced draft fans provide the driving force to move the cleaned gases to either the sulfuric acid plants or the elemental sulfur plant (when in operation) for removal of the sulfur dioxide.

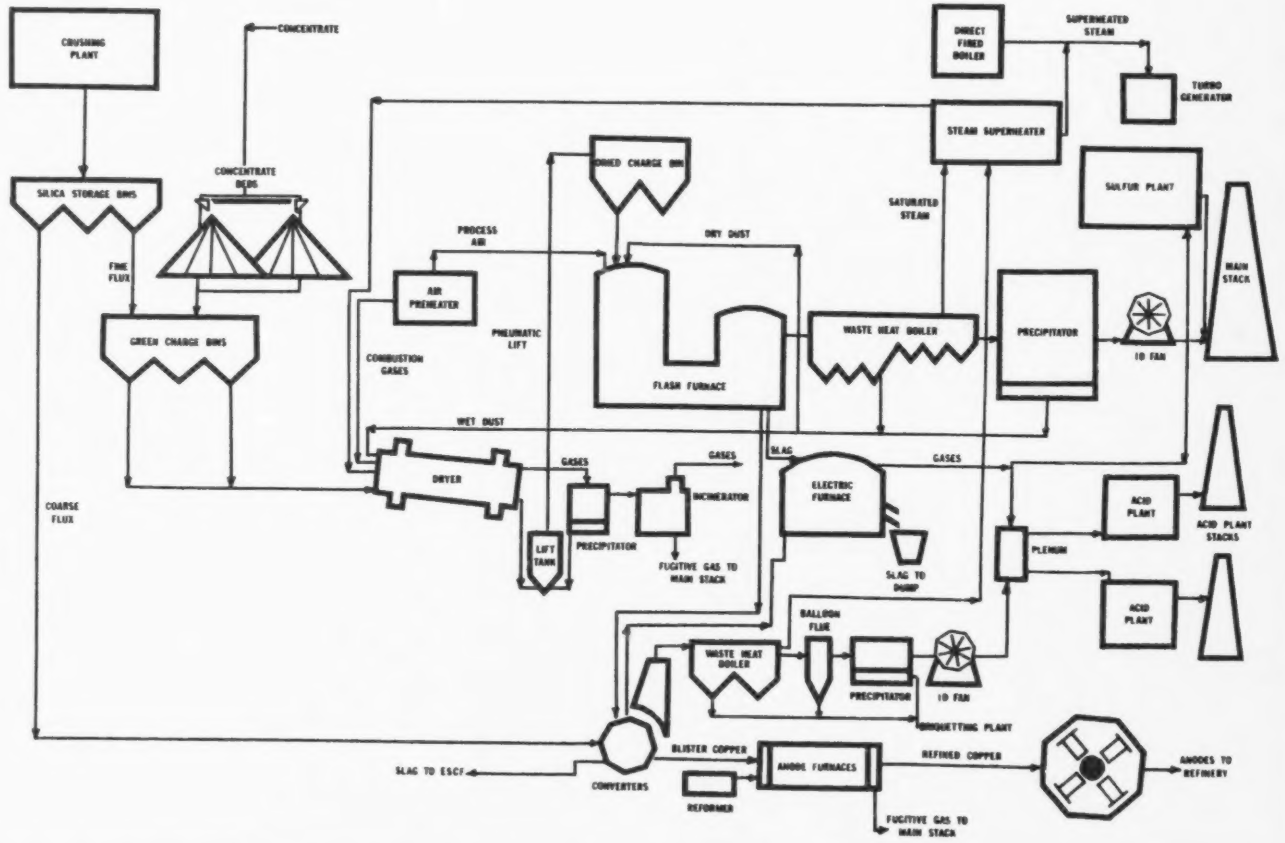
The flash furnace matte assaying about 60 percent copper is transported to the converters for further processing into blister copper. The anode furnaces ultimately refine the blister copper into anode copper which is cast into 750-pound anodes. Off-gases from each converter are cooled in a waste heat boiler to produce steam before entering two electrostatic precipitators in parallel for dust removal. The cleaned converter gases are mixed with the flash furnace gases in a mixing chamber prior to treatment in the sulfuric acid plants.

Flash furnace slag containing approximately 1.5 percent copper is skimmed continuously to an eleven megawatt electric furnace for cleaning of its copper content. Converter slag of about five percent copper is returned to the electric furnace for further processing. Final slag of about 0.7 percent copper is discarded in a designated area. Figure 1 shows the flowchart of the Hidalgo Smelter. The smelter was made operational June 1976 and the current daily production at the Hidalgo Smelter is 500 tons of anode copper and 2,000 tons of sulfuric acid.

Modern instrumentation including a computer is utilized at the Hidalgo Smelter for process control. An operator's console is provided at the central control room to permit the technicians to implement the computer control techniques.

FIGURE 1

HIDALGO SMELTER FLOW CHART


CRUSHING PLANT
CONCENTRATE
CONCENTRATE BEDS
SILICA STORAGE BINS
FINE FLUX
GREEN CHARGE BINS
COARSE FLUX
DIRECT FIRED BOILER
SUPERHEATED STEAM
TURBO GENERATOR
STEAM SUPERHEATER
DRIED CHARGE BIN
PROCESS AIR
AIR PREHEATER
DRY DUST
SATURATED STEAM
SULFUR PLANT
MAIN STACK
PNEUMATIC LIFT
COMBUSTION GASES
FLASH FURNACE
WASTE HEAT BOILER
PRECIPITATOR
ID FAN
WET DUST
DRYER
GASES
GASES
INCINERATOR
PRECIPITATOR
LIFT TANK
FUGITIVE GAS TO MAIN STACK
SLAG
ELECTRIC FURNACE
GASES
SLAG TO DUMP
PLENUM
ACID PLANT
ACID PLANT STACKS
ACID PLANT
BALLOON FLUE
WASTE HEAT BOILER
PRECIPITATOR
ID FAN
BRIQUETTING PLANT
SLAG TO ESCF
CONVERTERS
BLISTER COPPER
ANODE FURNACES
REFORMER
REFINED COPPER
ANODES TO REFINERY
FUGITIVE GAS TO MAIN STACK

Electrostatic Precipitators and Control of Particulate Emission

Electrostatic precipitators are installed at the Hidalgo Smelter to control particulate emissions. Electrostatic precipitator installations include:

- Three precipitators for the flash furnace off-gases
- One precipitator for the rotary dryer gases
- Two precipitators for the converter off-gases

The flash furnace precipitators and the converter precipitators were constructed in parallel to increase operational flexibility and to permit maintenance of one of the precipitators while the others are in operation.

Each flash furnace precipitator consists of a large metal casing with two continuous V-shaped hoppers at the bottom. The casing and hoppers are of all welded, gastight mild steel. Vertical baffles are installed in the hoppers to prevent any gas flow underneath the electrode systems. The hopper bottoms are welded to fit drag conveyors which remove the dust. Rappers are installed on the side of the hoppers to prevent dust accumulation.

Each precipitator for the flash furnace gases has four fields. One transformer-rectifier set is provided for each field. Proper gas distribution is accomplished by the installation of two gas distribution baffles at the inlet of the precipitator. The baffles are perforated stainless steel plates. A rapping mechanism is provided for the distribution baffles.

The internals of the precipitator consist of collecting plates and emitting plates which are made of stainless steel. The emitting plate has a peak type electrode where corona discharge takes place. The emitting plates are hung between collecting plates. A rapping mechanism is provided for the emitting system and the collecting system.

Actual operating experience of the flash furnace precipitators has indicated that the dust removal efficiency of this type of precipitator is in excess of 99.0 percent with inlet gases containing approximately 20 grains of dust per dry standard cubic foot and outlet dust grain loading of less than 0.03 grain per dry standard cubic foot of gas. The high efficiency is attributed to the following factors:

1. Proper gas distribution.
2. Plate-to-plate configuration on the collecting and emitting systems with appropriate guide bars resulting in proper alignment of the emitting electrodes with respect to the collecting plates.
3. Adequate rapping mechanism on both the collecting and the emitting systems.
4. Low resistivity of the flash furnace dusts.
5. Proper gas temperature as a result of continuous operation.
6. Proper maintenance of the precipitators.

The gases leaving the flash furnace precipitators are treated at the sulfuric acid plants.

The rotary dryer precipitator is similar in construction to the flash furnace precipitator with the following exceptions:

1. A "plate to wire" configuration is employed.
2. The plates are made of mild steel.
3. The precipitator casing has a flat bottom with two scrapers to remove the collected dusts.
4. The precipitator has three fields with related transformer-rectifier sets.

Operation statistics for the dryer precipitator have demonstrated that its dust removal efficiency is in excess of 99.0 percent. The precipitator is subject to an inlet dust grain loading of about 100 grains per dry standard cubic foot. Dusts collected are returned to the pneumatic lift for transportation to the flash furnace for smelting. Gases leaving the precipitator contain less than 0.03 grains of dust per dry standard cubic foot. These gases are discharged into the atmosphere through a 600-foot stack.

The converter precipitators utilized at the Hidalgo Smelter are of plate to wire configuration for the collecting and emitting systems. The precipitator is made of a mild steel casing and has a single mild steel hopper with screw conveyors at the bottom. The precipitator has three fields with related transformer-rectifier sets.

The efficiency of the converter precipitators ranges from 85 percent to 90 percent. Low efficiency is attributed to:

1. High resistivity of converter dust causing degradation in precipitator efficiency.
2. Intermittent operation of converters causing wide fluctuation in operating temperature which ultimately results in excessive corrosion.
3. High adhesiveness of dust causing buildups on collecting electrodes and short circuiting.

The gases leaving the precipitators are treated in the sulfuric acid plants.

Table I indicates some typical operating parameters of the various precipitators.

Maintenance on precipitators has high priority at the Hidalgo Smelter. A maintenance team of six persons is assigned for the routine cleaning and maintenance of the precipitators in order to increase the equipment availability and to maintain efficiency.

Table I: Comparison of Precipitator Performances

Precipitator Application	Primary Amperage AC-Amperes	Load Amperage DC-Amperes	Voltage DC-Kilovolts	Efficiency %	Availability %
Flash Furnace	100	0.5	40	99.0	85
Rotary Dryer	70	0.3	40	99.0	95
Converters	50-75	0.25	40	85.0	70

Baghouses are also used at the smelter to control dust emissions from the following smelter activities:

-Crushing plant operation.

-Transportation of concentrates. Baghouses are installed at transfer towers at various locations.

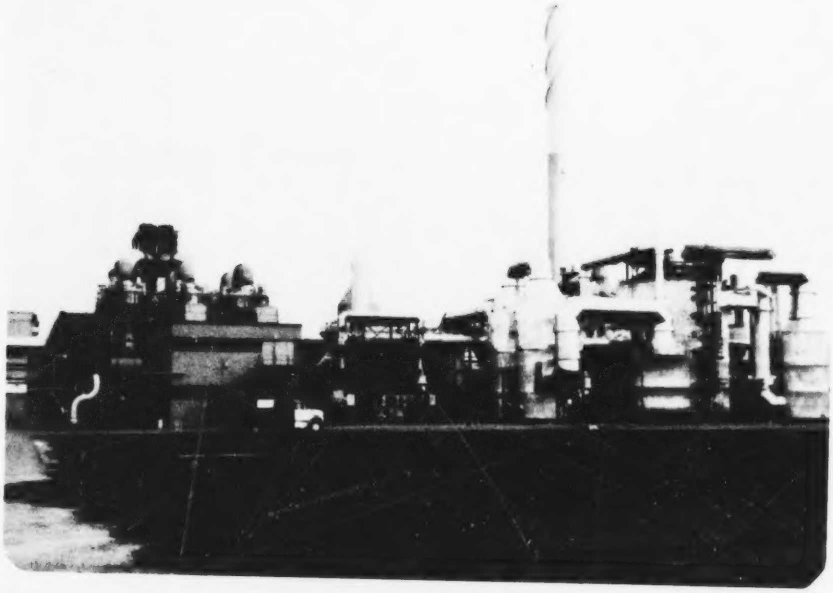
-Transportation of fine reverts to the electric furnace.

The maintenance team is responsible for changing bag filters and other routine maintenance on the baghouses. There are thirteen baghouses in use at the Hidalgo Smelter.

Operation of the Sulfuric Acid Plants

The Hidalgo Smelter was designed originally for the flash furnace gases to be treated in an elemental sulfur plant and the converter gases to be treated in a sulfuric acid plant with capacity of 1,180 tons per day. However, energy costs have made it uneconomical to operate the sulfur plant. Therefore, a second sulfuric acid plant of 2,200 tons capacity per day was constructed. Both plants treat the mixed flash furnace gases and converter gases. Gases from the flash furnace and the converter are mixed in a mixing chamber. The combined gases of approximately eight percent SO_2 can be routed to either or both acid plants for production of sulfuric acid. With the flash furnace operating at its capacity of 2,000 tons of concentrate per day, gas volume is approximately 190,000 SCFM with 8.0 percent SO_2. The original sulfuric acid plant is designed for 90,000 SCFM at 7.3 percent SO_2 while the second sulfuric acid plant can handle a maximum of 129,000 SCFM containing 8.3 percent SO_2.

Figure 2 shows the sources of the smelter gases and their treatment in the sulfuric acid plants.



Sulfuric Acid Plant

Both plants are double absorption sulfuric acid plants. Each acid plant contains a gas cleaning section and a double absorption section. The smaller acid plant employs a gas scrubbing/cooling tower, four lead mist precipitators as equipment in the gas cleaning section while the larger sulfuric acid plant utilizes two venturi scrubbers, five lead "star coolers" and six plastic/lead mist precipitators as equipment for the gas cleaning section. The scrubbers and coolers adiabatically cool the incoming gas from about 500° F to about 90° F. The majority of the dusts and sulfur trioxide in the incoming gases are removed from the gas stream in the scrubber to form weak sulfuric acid and sludge. The remainder of the dusts and sulfuric acid mists are removed by the mist precipitators. The gas leaving the gas cleaning section is essentially clean but saturated with moisture.

The SO_2 blowers draw the gases through a drying tower countercurrent to the circulating 93 percent sulfuric acid. Moisture in the gas is absorbed by the sulfuric acid. The dried gases pass through a mesh pad filter to remove droplets of acid prior to entering the SO_2 blowers that deliver the gases to a catalytic converter through a series of shell and tube heat exchangers. The 93 percent sulfuric acid is diluted by the absorption of moisture. However, its concentration is restored by cross-flow of 98 percent acid from the absorbing towers.

In the catalytic converter, the sulfur dioxide is converted to sulfur trioxide in the presence of vanadium pentoxide in four catalyst passes.

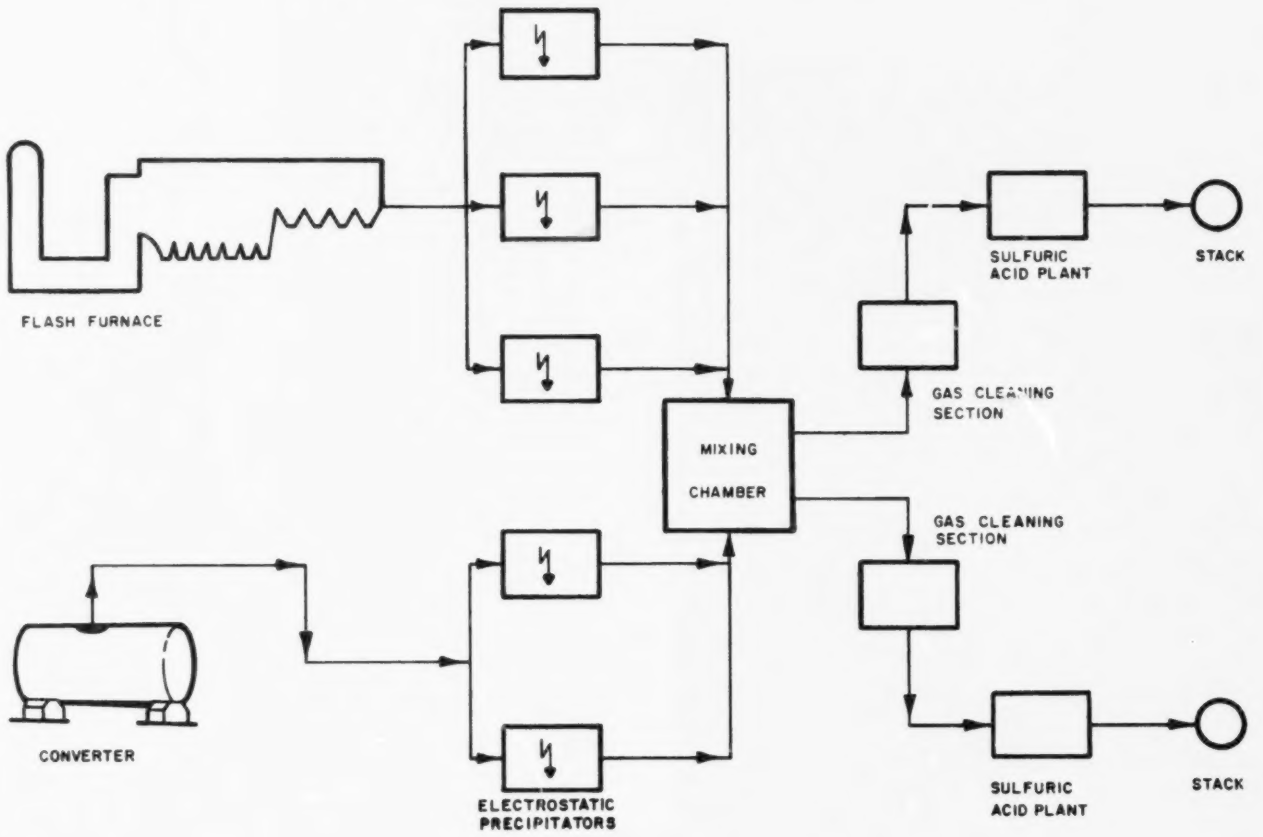
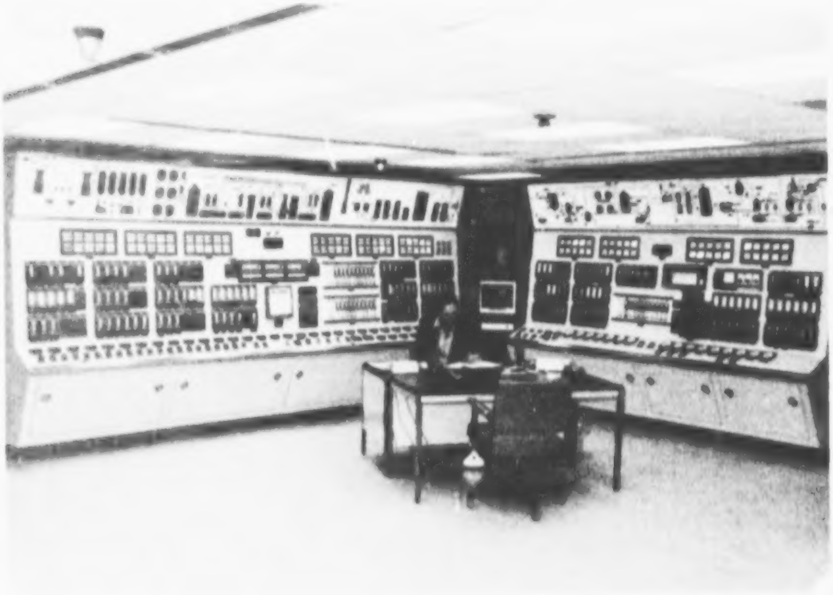

FLASH FURNACE
CONVERTER
ELECTROSTATIC PRECIPITATORS
MIXING CHAMBER
GAS CLEANING SECTION
GAS CLEANING SECTION
SULFURIC ACID PLANT
SULFURIC ACID PLANT
STACK
STACK

FIGURE 2
TREATMENT OF SMELTER GASES IN THE SULFURIC ACID PLANTS

Oxygen to sulfur dioxide ratio is controlled by a process computer to allow proper conversion of sulfur dioxide to sulfur trioxide. Normal conversion efficiency is in excess of 99 percent. The sulfur trioxide from the second pass is absorbed in the primary absorbing tower by 98 percent acid, and the sulfur trioxide from the fourth pass is absorbed by 98 percent acid in the final absorbing tower. Acid strength in the 98 percent acid circuit increases constantly due to absorption of sulfur trioxide. However, the acid strength is maintained by cross-flow of 93 percent acid from the drying circuit or by addition of water. Heat generated by dilution in the 93 percent acid circuit and by absorption in the 98 percent acid circuit is removed with cooling water by means of shell and tube heat exchangers.

The tail gases pass through high efficiency "candle filters" installed at the upper portion of the final absorbing tower. Gases leaving the acid plants contain less than 650 ppm SO_2 . The two sulfuric acid plants enable the Hidalgo Smelter to meet the 90 percent sulfur capture requirement imposed by New Mexico's air pollution control regulations applicable to new copper smelters.

The control room for the sulfuric acid plants is equipped with modern analog controllers for automatic control. The computer control loops enable the sulfuric acid plants to achieve a high degree of productivity and maximum sulfur recovery.



Sulfuric Acid Plants Control Room

The operation of the sulfuric acid plants is very energy intensive. Large amounts of electrical energy are required to operate two 5000-HP motors in the larger acid plant and two 3000-HP motors in the smaller acid plant. In addition, numerous pumps of various sizes are required to circulate the sulfuric acid through the towers and to circulate cooling water through the acid coolers. Fuel oil is consumed to operate the preheaters in order to maintain operating temperature at the catalytic converter during periods of temporary inactivity and prior to startup following a shutdown. Experience at Hidalgo indicates that the average energy consumption is 2.7 MMBTU per ton of sulfuric acid produced. Approximately 40 percent of the total energy consumed by the entire smelting complex is attributable to pollution abatement requirements.

Conclusion

With the installation of the high efficiency electrostatic precipitators and the double absorption sulfuric acid plants, the Hidalgo Smelter is able to comply with environmental regulations regarding particulate emission, sulfur dioxide emission and overall sulfur capture. At the Hidalgo Smelter, particulate emission is less than 0.02 grain per dry standard cubic foot of gas as compared with the applicable State regulation of 0.03 grain per dry standard cubic foot of gas. Sulfur dioxide emission at the sulfuric acid plants is less than 650 ppm while the overall sulfur recovery is above the 90 percent requirement.

ON THE ELECTROSTATIC PRECIPITATION OF COPPER SMELTER EMISSIONS

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ABSTRACT

The electrostatic precipitation of high concentration, low resistivity dust is primarily limited by two mechanisms: corona quenching and its effect on the electrohydrodynamic secondary flow, and the restitution and subsequent migration of charged particles from the EP electrodes. These mechanisms are discussed as they apply to the conventional Cottrell geometry and to a modified two-stage EP design. Pilot test results from a copper smelter application are presented to illustrate electrostatic precipitation in the latter electrode arrangement.

INTRODUCTION

Particulate emissions from copper smelting can be collected by the electrical precipitation process and good tools for predicting EP performance from physico-chemical data are beginning to emerge[1]. Although considerable effort has been expended towards an understanding of the particulate collection problem, particulate collection does not appear to be the primary concern of suppliers and users of pollution control equipment in the metallurgical industry. The central problem lies in the control of the hostile components of the stack gas. These condensibles plague particulate control systems through the deposition of strongly adhering dusts, corrosion, and the secondary disposal of toxic substances. Problems associated with these aspects of pollution control are not as predictable as particulate collection, and thus pilot testing often becomes necessary.

The goal of the present paper is to affirm the necessity of pilot testing for each pollution abatement or process material recovery application. This theme is echoed from the aisles of a recent USEPA conference on Particulate Collection Problems in the Use of Electrostatic Precipitators in the Metallurgical Industry (Denver, 1977). The paper begins with a discussion of the electrostatic precipitation process in general and then proceeds to a review of some recent work done at United McGill on electrohydrodynamic effects in electrostatic precipitators. Finally, some recent exploratory investigations at a copper smelter facility will be reviewed. Although the subject application constituted a pilot scale investigation, it is worth sharing as an amplified example of some traditional "EP" problems experienced in the copper smelting industry.

FUNDAMENTALS

Electrostatic precipitation employs the electrical force, between charged bodies, to separate particulate matter from a gas stream. The electrical force, which is responsible for the precipitation, arises from opposing charge accumulations on separated EP electrodes. This charge separation is accomplished by a high potential dc power supply (transformer and rectifier and possibly auxiliary pulsing equipment) with its terminals connected to the EP electrodes.

The electrical precipitation process is simple: Particulate matter is charged, then transported to a collection surface by an electric field, and then removed to an external receptacle. For the efficient operation of an EP, the order of these events cannot be changed.

Application of the electrical precipitation process to physical situations is not simple. There are several charging mechanisms, migration and electrohydrodynamic effects, and adhesive mechanisms which interact to determine the performance of an electrode geometry in a given application.

The most commonly discussed charging mechanism in electrostatic precipitation is the dc corona[2]. Other gaseous charging schemes, under laboratory or pilot scale investigation, include pulsed corona[3,4,5], ion beam systems[6], and radioactive chargers[7]. Each of these devices serves the same purpose -- gaseous ion charging. Within a few tenths of a second after entering the charging region, a particle gains a charge near the Pauthenier limit.

$$Q = \frac{3\ \varepsilon}{\varepsilon+2} \quad (4\pi\varepsilon_o E_o a^2) \tag{1}$$

where ε is the dielectric constant for the particle, ε_o is the permittivity of vacuum, E_o is the static field at the particle, and a is the radius of the particle.

When a particle lies on an electrode surface, it gains a charge by induction. The charge gained by a particle, exposed to this second charging mechanism, is near that which the particle would have gained by the gaseous charging mechanism (1). This charge is called the Maxwell or Felici charge[8].

$$Q = \frac{\pi^2}{6} \quad (4\pi\varepsilon_o E_o a^2) \tag{2}$$

The charging time, corresponding to this mechanism, is on the order of the permittivity-resistivity product -- microseconds for many metallurgical dusts.

Inductively charged particles (almost all electrostatically deposited particles) tend to detachment unless resisted by strong adhesive forces. The repulsion force on the particles is on the order of

$$F = 1.37 \quad (4\pi\varepsilon_o E_o^2 a^2) \tag{3}$$

As a particle becomes embedded in a surface, its charge and detachment tendency decrease to zero.

A third charging mechanism is a solid state effect, which occurs when particle surfaces of differing composition come into contact[9]. These contact effects result in a small charge transfer which depends on the interfacial capacitance between the materials and on the difference between their surface state potentials. Although less significant as a particle charging mechanism, the charge transfer due to non-uniform composition results in very strong adhesive forces in electrostatically deposited dust layers[10]. The natural charge of dispersed dust obtained its charge from this mechanism.

Another charging mechanism is the microdischarge process, or corona discharge, of a particle as it approaches an electrode surface[11]. This mechanism leads to charge reversal and difficult collection conditions for low resistivity particles.

Finally, in real situations, the corona is bi-ionized. That is, it contains ions of both polarities. Streamers and back ionization, both of which contribute ions of the opposite polarity from those of a simple dc corona, are exhibited in Fig. 1. At present, a number of investigators are looking at electrostatic precipitation in bi-ionized systems.[12, 13, 14]

In addition to its charging function, the corona induces secondary gas flows on the order of 1-2 m/s in the vicinity of the grounded electrodes and on the order of 15-20 m/s in the vicinity of the corona electrodes[15]. These flows transport particles, irrespective of their charge, to the boundary layer near the collection surfaces where they must collect electrostatically. Particles which entrain and/or are of opposite polarity continue with the return flow or migrate to the corona electrode. Recently, T. Yamamoto succeeded in numerically solving the Navier-Stokes equations for the electrohydrodynamic field of a positive polarity EP of the Cottrell design[16]. Although limited to the particle free conditions of no cross flow, the results confirm the flow patterns he observed using Schlieren photography. His model is presently being extended to include the cross flow components.

The choice of EP electrode system requires a consideration of all these mechanisms as well as a concern for dust adhesion and removal. The many designs currently employed in the metallurgical industry can be divided into single and two-stage geometries. The Cottrell or wire in duct geometry employs a series of wires (or rods) to establish a corona and to collect the particles. To serve this dual role, a high-potential difference is required (50KV). Low resistivity particles, which easily reverse their charge (often before they reach the surface) and require some tenths of a second to recharge in a corona, are thus difficult to collect in such a geometry. Fortunately, most metallurgical dust adheres to the electrode surfaces on one of the first few impacts with a surface and thus collection is achieved.

In two-stage EP's, a parallel plate collection field is employed to collect particles of either polarity. The potential difference between the electrodes in two-stage units is lower than that for single stage units because it only needs to serve one role, to establish a collection field. One loses the local electrohydrodynamic flows, but gains more collection area per unit EP volume and a lower electrode power consumption. The role of the corona in two-stage EP's is to provide the initial particle transport to the collection electrodes.

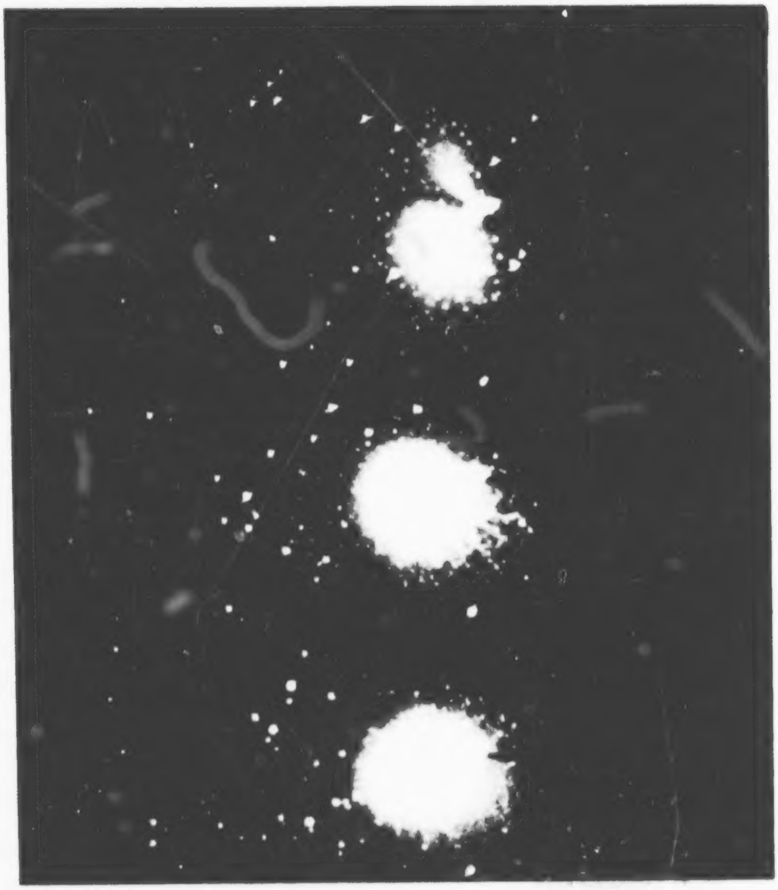

Fig. 1. Positive dc corona from a linear array of needles. (Note pre-breakdown streamers and reverse ionization.)

An industrial scale, modified two-stage EP design is exhibited in Fig. 2. This design was employed in the pilot testing program for two reasons: 1) The modified two-stage design was found to perform well on two other high concentration dusts (Iron oxide 25 gm/Nm^3, Alumina 9 gm/Nm^3) with similar resistivity in the EP operating range, and, 2) There was an interest in determining the feasibility of copper separation from fluid-bed roaster emissions. From a research standpoint, successful collection of high concentration dust is an indication of the lifetime for charge on particles suspended in a bi-polar charge system.

About a year ago, C. Noll and T. Yamamoto observed particulate collection in this geometry using mist particles[14]. In Fig. 3, a mineral oil/CO_2 mist was injected as a tracer with the gas stream. The electrodes were operated near sparkover potential (20kV). The gas speed was near 0.65 m/s. Note that the particles were propelled to the collection electrode surfaces. At a much lower potential (13.5 kV), yet still above corona onset, a similar particle transport was observed (See Fig. 4). In this case, the mist was oil-soaked cigar tobacco fumes. The gas speed was 1 m/s. There are two tracer streams in the photo. A further experiment employing Schlieren photography, revealed vortex formation near the grounded electrode surfaces (See Fig. 5). Similar investigations were carried out for the Cottrell geometry. Gas velocity/pressure studies, using hot wire anemometry and micromanometer techniques were also performed to support the numerical studies mentioned above.

The interest in this work was motivated by the casual and almost universal observation of dry particulate collection on both polarity electrodes in EP's (See Fig. 6). It was concluded that both polarity particles can, and do, normally exist in the electrode region of an EP and that the role of the corona in two-stage designs is to effect initial particle transport to the grounded collection surfaces. This was confirmed by field experience which indicated a rapid rise in efficiency with corona onset to some steady state level determined by entrainment -- See Figs. 7 and 8. To obtain a greater efficiency, a larger EP system would be needed.

One of the traditional problems of the implementation of two-stage precipitators to industrial processes lies in corona quenching[17]. The current drawn by an EP electrode system is composed of two parts: ionic and particulate.

$$J \propto c_p K_p E + c_i K_i E \tag{4}$$

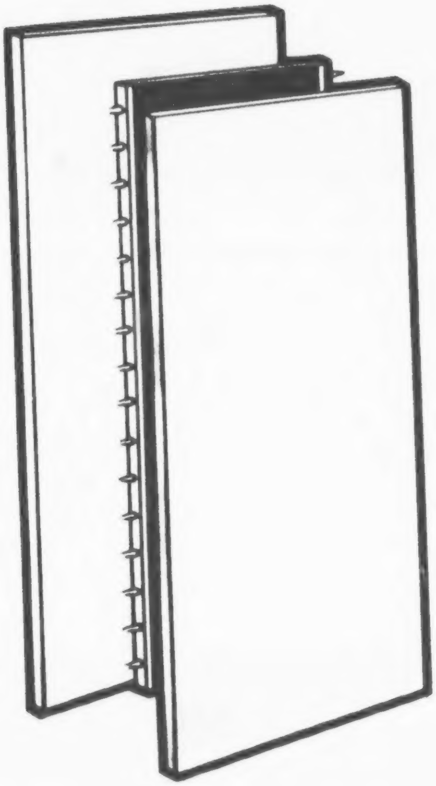

Fig. 2. Modified two-stage EP design. (Needle-plate in duct geometry (NPD).)

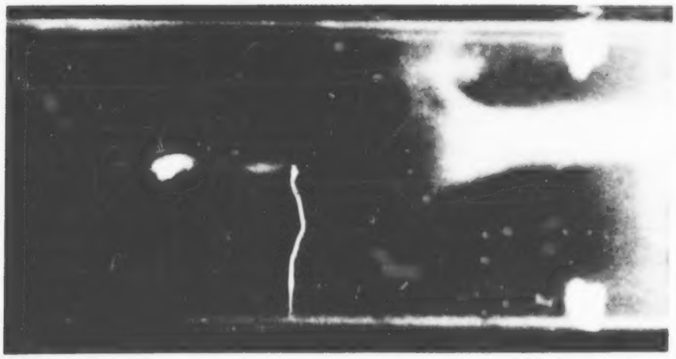

Figure 3. Mist collection in NPD geometry (Potential at sparkover 20 kV).

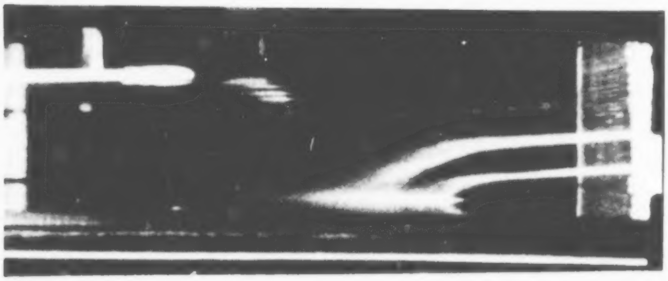

Figure 4. Mist collection in NPD geometry (Potential below sparkover 13.5 kV).

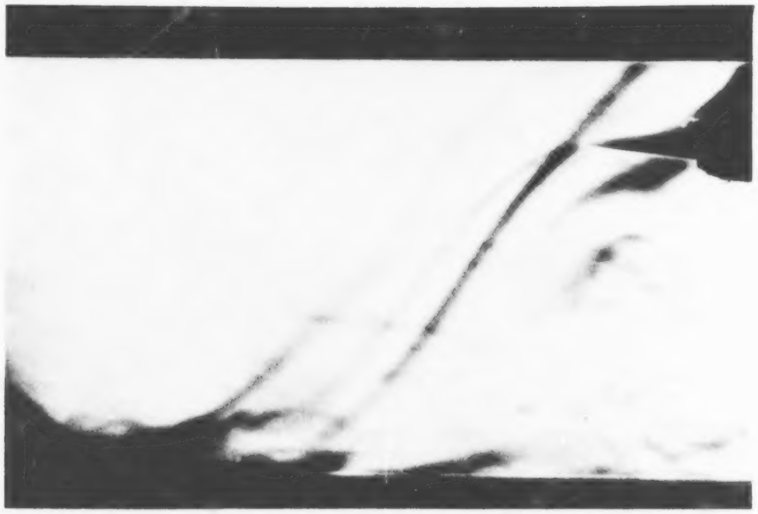

Figure 5. Electrohydrodynamic secondary flow in needle to plane geometry. Schlieren photograph recorded with needle shank parallel to plane.

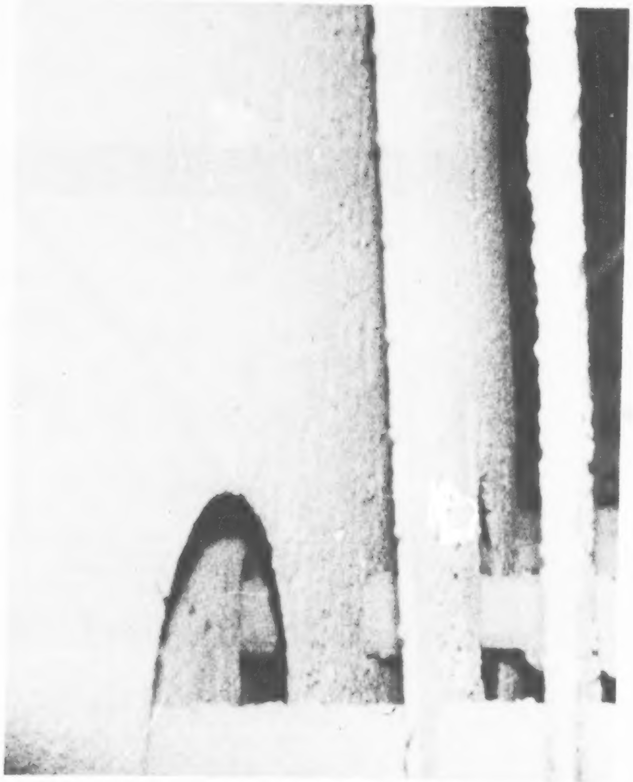

Figure 6. Dry particulate collection on both polarity electrodes of NPD geometry.

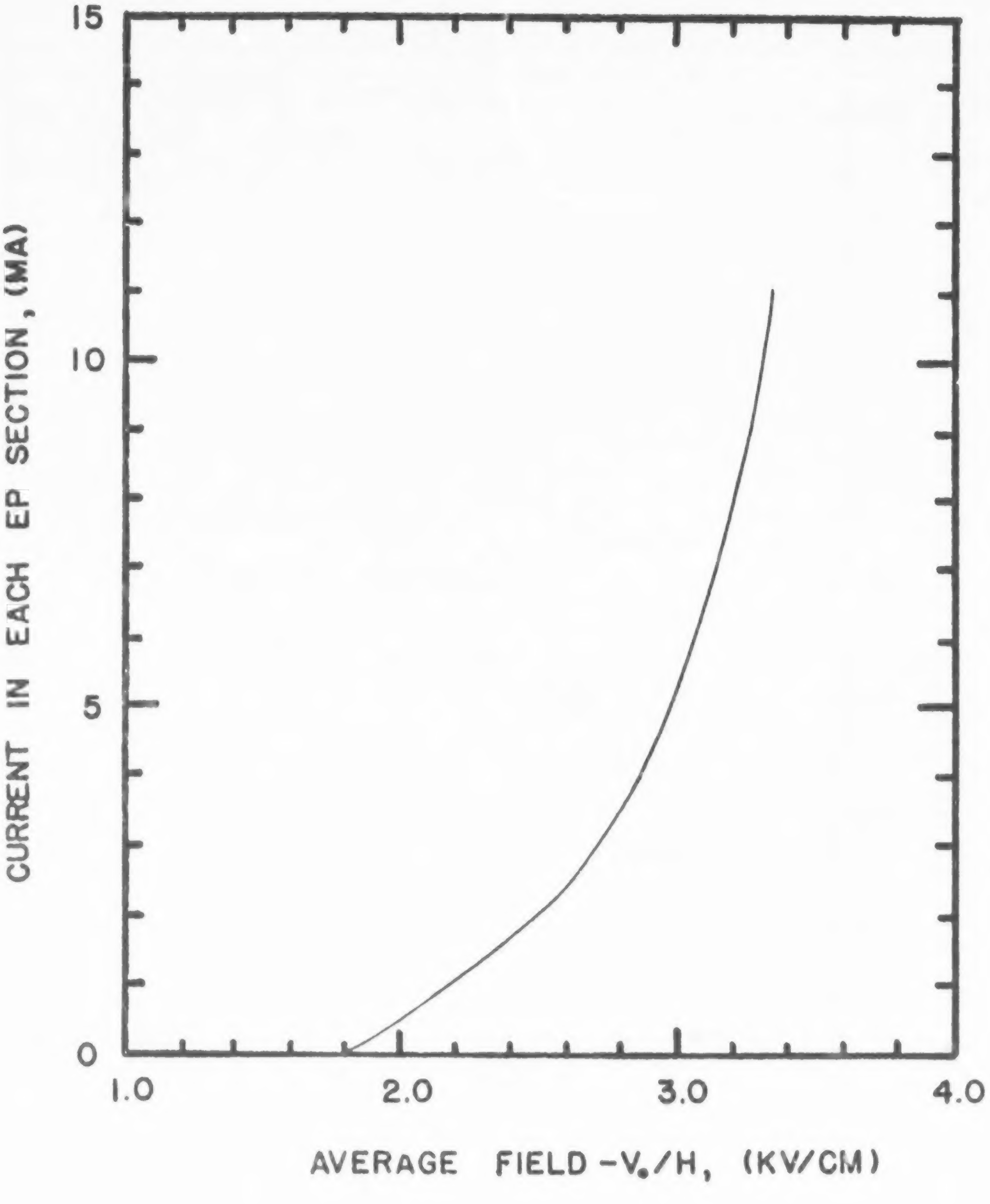

Figure 7. Corona current vs. mean field data for pilot unit.

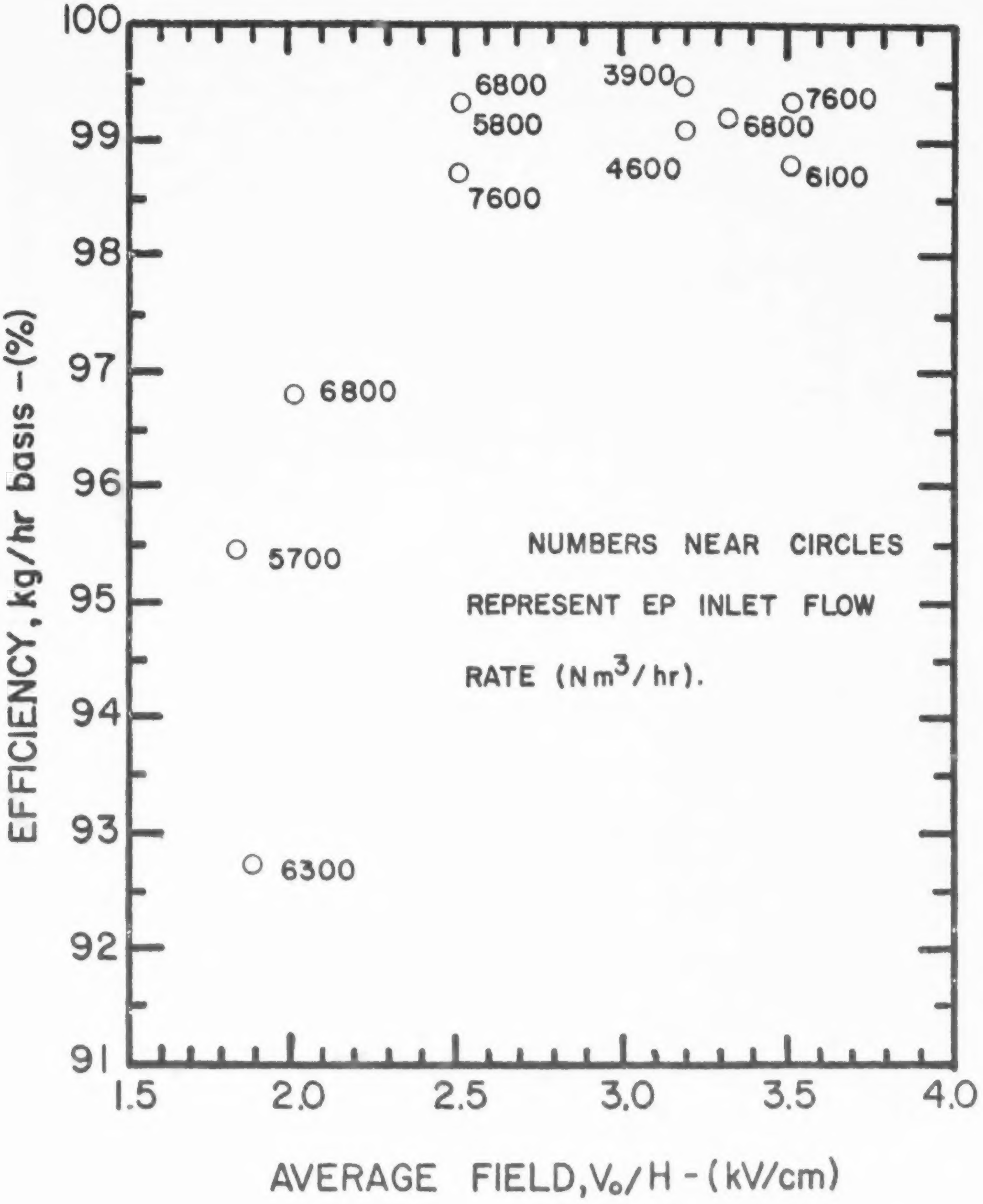

Figure 8. Efficiency vs. mean field corresponding to current data in Figure 5.

where c, K, E represent the concentration mobility, and static field for the particle and ion systems (subscripts), respectively. When the concentration of particles is high or the dust is very fine, attachment of the ion concentration to the particulate system leads to corona quenching. The particles have a far lower mobility than the ions and thus the field near the corona electrode is suppressed or bucked. Consequently, the secondary flow of the corona is suppressed. Although significant corona quenching was observed in the first field, on the BOF application (Iron oxides), it did not penetrate the second or later fields and impair the systems collection. It was thus proposed to test this modified two-stage design on a pilot scale smelter application. This pilot work would extend the useful concentration range of this geometry to at least 75 g/Nm^3.

PILOT INVESTIGATION

a. Description of Pilot facility

At the copper smelter, the primary objective was to separate copper compounds and elemental copper from the emissions of a natural gas fired, fluid-bed roaster. It was the contention that if the EP inlet temperature was maintained at a sufficiently high level, volatilized arsenic and other condensibles would pass through the precipitator, while solid copper emissions would be collected[18].

A temporary 0.51 m diameter duct system, with dampers, was erected, allowing a portion of the flue gas from the roaster to enter the pilot precipitator (See Fig. 9). The total length of the duct was approximately 40 m and was insulated. A cyclone precleaner was installed 9 m upstream of the precipitator.

The EP exhaust fan was then used to control the volume of flue gas from the existing emission system, with the EP's horizontal evaporative cooler used to control the flue gas temperature when necessary. The complete specifications of the pilot precipitator installation is summarized in a reference[19].

The test schedule centered around running the precipitator at the highest attainable inlet temperature while altering the precipitator gas flow-through velocity.

b. Description of Sampling Procedure

Emission testing was accomplished by conventional EPA Method 5, as outlined in the Federal Register[20], with several

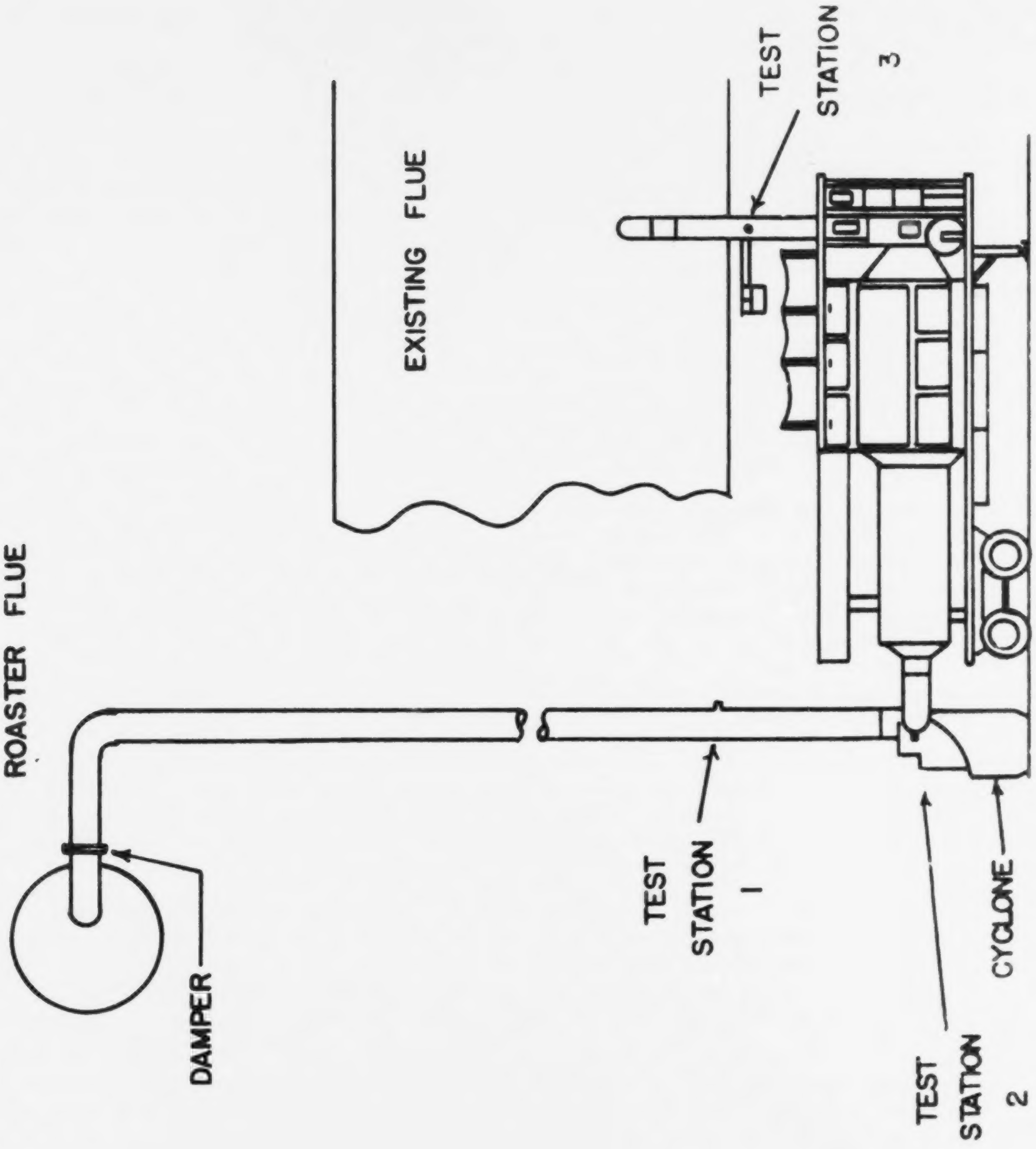
ROASTER FLUE
EXISTING FLUE
TEST STATION 3
DAMPER
TEST STATION 1
CYCLONE
TEST STATION 2

Fig. 9. Functional schematic of pilot electrostatic precipitator.

exceptions. All testing was performed using an in-stack Alundum filter ahead of the standard sample train hot box filter. The filter, filterholder, and stainless steel nozzle were preheated to stack temperature, with the filter gasket tightened before each test. The in-stack filter was contained entirely inside the test duct and prevented the traversing of the stack as outlined in EPA Method 5. As a result, testing was performed at the duct center line, with a center line coefficient applied to the results.

Simultaneous measurements were secured at the inlet and outlet of the precipitator and at the inlet to the cyclone pre-cleaner. Also, transmissometers, mounted on the precipitator inlet and outlet, were employed to keep a 24-hour record of the inlet and outlet opacity.

The probe unit was attached to a pyrex filter holder, containing one Gelman "A-E" glass fiber filter. The filter holder was maintained at 395K (250°F) during tests and was connected to the first of four impingers.

All four impingers were of a modified Greenburg-Smith design, reducing pressure loss across the sample train. The first three impingers contained 250 ml, 100 ml and 100 ml respectively, of 30% sodium hydroxide. The fourth contained 500 g of dry silica gel. All four impingers were placed in an ice bath to prevent the sample from rising above 295K (70°F) in the last impinger.

The liquid in Impinger 1, the combined liquid in Impingers 2 and 3, and the impinger washings were measured and placed in three separate polyethylene bottles. The silica gel in Impinger 4 was weighed immediately to determine its weight gain. The two filters (in-stack and out-of-stack) were placed in watch glasses, over dried at 105°C, desiccated and weighed. The in-stack and out-of-stack portions of the sampling probe were washed with water into two separate polyethylene bottles.

The filter particulate was dissolved with nitric acid and analyzed along with the other solutions. The in-stack and out-of-stack washes were evaporated at 378K (220°F), desiccated and weighed. Weights and solution volumes were reported for each sample, along with the assays for copper, arsenic, zinc, bismuth, selenium, tellurium, lead and antimony. The filter catches, plus the evaporated probe washes, represented the dry particulate collected. All chemical analyses were performed in the sponsor's laboratories.

c. Discussion of Results

Tests on the gas-fired fluid-bed roaster were characterized by both high particulate loading and very high moisture content This combination caused problems in the existing system by frequently plugging the cyclone pre-cleaners, and was a particular problem if the system was allowed to cool after a period of down time. Condensation during start-up also resulted in cyclone plugging.

The pilot's EP's horizontal evaporative cooler was used to obtain test data in the EP inlet temperature range from 590K (600°F) to 645K (700°F). No tests at temperature below 590K were performed, as the EP outlet temperatures were deemed too low for useful arsenic separation information to be generated[18].

Particulate collection efficiency across the EP was maintained at high levels, with copper recovery rates as high as 99% on an inlet to outlet kg/s basis (See Table I). Total particulate collection efficiencies were also at this high level. As a result, the EP flow through velocity was controled at average gas speeds in the range from 0.75 to 1.1 m/s. These tests, plus tests with only one and two EP fields energized provided additional design data as well as arsenic separation information. A total of 27 EPA Method 5 tests were secured from various points along the roaster duct system.

Method 8 test results indicated extremely high sulfuric acid and sulfur dioxide concentrations in the gas stream: 0.5 $\times 10^4$ mg/Nm^3 SO_2. In addition, tests across the cyclone indicated that its efficiency was low at the inlet velocity of 9.0 m/s.

Heavy flue gas dust loading and high moisture content produced several problems. During EP startup, high moisture condensation was observed. The initial hopper and cyclone discharge dust was of a thin muddy consistency, and during this period, the cyclone would easily plug. To circumvent this problem at the EP, the EP hopper rotary valves were kept in continuous operation.

Heat loss through the system was also a problem. Due to height limitations, while transporting the Mobile Precipitator, the high potential insulators were located inside the main EP chamber. A flow of clean purge air was used to prevent the low resistivity dust from building up on the insulators and causing shorting or low potential arcing.

Initially, this flow of air was kept very low and several tests were completed. However, the insulators later became coated with dust, which was not acceptable, and the volume of purge air had to be increased. After this adjustment, the precipitator ran well and continued to effectively collect particulate emissions. Unfortunately, as a result of the increase of insulator purge air, the EP outlet temperature showed a significant decrease.

Dust fallout in the duct was evident with gas flow velocities of 8.0 - 12.7 m/s. Twice during the testing program, the inlet duct filled halfway with dust, requiring that the precipitator be shut down and the duct cleaned.

TABLE I. Typical Mobile EP Performance Data

(Gas-fired fluid-bed roaster)

Average EP velocity	1.13 m/s	(3.70 ft/s)
Gas Volume	2.49 m^3/s	(5279 ACFM)
Moisture	43%	
Inlet loading	1.96 kg/s	(259 lb/hr)
Efficiency (kg/s at outlet vs. inlet basis)	98.9%	
EP Field potentials	26/18.5/23 kV*	
EP Field currents	2/5.7/4.2 mA	
Total Energy (Power) supplied to the EP fields	554 joules/Nm^3	(243 watts/ 1000 SCFM)
EP inlet temperature	620K	(653°F)
EP outlet temperature	500K	(445°F)
Cu recovery	99.0%	
As recovery	68.6%	

* For the high dust loading case, the plate spacing in the first field was increased from 8 cm to 10 cm.

CONCLUSIONS

The following conclusions were drawn from the Mobile Precipitator pilot testing program and the fundamental studies.

1. The particulate emissions from the gas-fired, fluid-bed roaster at this copper smelting site were easily collected, as evidenced by a consistent 97-99% recovery rate for the three field mobile pilot unit. This conclusion was further based on similar experiences with the same mobile unit on over 25 other applications[19].

2. The separation of arsenic and copper was observed during the Mobile EP testing program, but the results were not definitive because of the temperature drop across the EP. The potential for success with an EP operating with an inlet temperature near 700K (800°F) and an outlet temperature near 645K (700°F) remains only speculative. Cold walls and unintentional ambient air leakage lowered the Mobile Precipitator temperature below that which was desired. During the testing program, the ambient temperature was typically below 273K (0°F). Additional testing with a heated shell precipitator is recommended for obtaining more representative data.

3. Corona quenching is indicated in the data of Table I by a suppression of the first field current. However, in a test where only the first field was energized, 64% of the inlet loading was shown to be collected in that field. These results demonstrate that the modified two-stage geometry would successfully collect dust under conditions of corona quenching.

ACKNOWLEDGEMENT

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SPECIAL CONTROL SYSTEMS

Session Chairman: Grady B. Nichols
Southern Research Institute
Birmingham, Alabama

APPLICATION OF COTTRELLS IN ASARCO'S NONFERROUS SMELTERS

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Abstract

For the past 66 years ASARCO has designed and built cottrells for particulate recovery in our various copper, lead, and zinc smelters. The ASARCO design has advantages and disadvantages when compared to commercially available units for dry application. Data on inlet and outlet grain loading and particulate size distribution have been accumulated for both roaster-reverberatory and converter cottrells. ASARCO recognizes the effectiveness of gas conditioning with respect to the control of particulate resistivity. Conditioning agents used are: water, sulfuric acid, SO_3, and hydrated lime. Some ASARCO plants use only one agent while others use a combination of two or more. Measurements have been made that show a marked improvement in particulate and SO_3 recovery from a gas stream when water conditioning was used.

Introduction

The involvement of ASARCO Incorporated with the application of cottrells at nonferrous smelters began nearly 71 years ago. For the past 66 years the experience has included the use of ASARCO's own designed cottrells. The company exposure to operating cottrells began in July 1907 when Dr. F. G. Cottrell installed the first commercially successful cottrell in the United States, at ASARCO's Selby, California lead smelter.(1,3) This unit, as described by F. G. Cottrell, was made up of several rows of four inch wide by four feet long lead plates installed vertically, on four inch centers, within a four foot by four foot lead flue. Lead covered iron rod discharge electrodes with mica, which gave a point discharge for the current, were placed between each pair of plates. Much time and effort were expended in experimentation to develop the ideal discharge electrode for the Selby unit. The design that was accepted as the ultimate was of lead impregnated with mica.(2)

Generally speaking, cottrells we have built in ASARCO are categorized on the basis of the design of the collecting electrode. ASARCO's experience in dry cottrell operation over the past 66 years has been directed to four basic designs of collecting electrodes, namely; pipe, corrugated plate, wire screen, and wire cage.

ASARCO started to design and build cottrells for our own smelters in 1912 when the first large multiple-pipe precipitator was built at our Garfield smelter near Salt Lake City, Utah.(3,4) ASARCO installed only pipe, screen, and wire cage type cottrells in the period between 1912 and 1924. After this date, all our dry cottrell installations have been of the corrugated plate design.

Many of the earlier design cottrells have been abandoned and Table I lists those currently installed in ASARCO's lead, zinc, and copper smelters. Two cottrells listed, for the El Paso lead sinter and the Tacoma anode furnace operations, are scheduled to start this year. All the cottrells in Table I are of the plate design except the Tacoma reverberatory installation which consists of a pipe and plate cottrell in series. Eleven of those tabulated are ASARCO design and two, the Columbus and the Hayden converter cottrells, are commercially available.

Table I
Cottrell Operations in ASARCO

Location	Operation	Conditioning	Gas Temperature °C
Columbus	Zn. Roaster	None	380
Corpus Christi	Zn. Roaster	None	300
East Helena	Pb. Sintering	Water	225
El Paso	Pb. Sintering	Water	230[a]
El Paso	Cu. Converter	Water-Acid	138
El Paso	Cu. Roaster	Water-Acid	116
El Paso	Cu. Reverberatory	Water-Acid	116
Hayden	Cu. Converting	Water	230
Hayden	Cu. Roaster-Reverberatory	Water	125
Tacoma	Cu. Reverberatory	Water-Acid	100
Tacoma	Cu. Converter	Water-Lime	138
Tacoma	Cu. Converter	Water-Lime	138
Tacoma	Cu. Anode Furnace	Water[a]	93[a]

[a] Anticipated

Design Features

The cottrells we are building today are essentially the same design used in the mid 1920's. Over the years modifications have been made that result in improved performance. Insofar as I have been able to determine, we have always used corrugated plate composed of iron alloyed with 0.2% copper. Rows of corrugated plates are spaced at 6" centers. Round wires have been used as discharge electrodes since 1912.(4)

The number of gas passages per field and the active lengths of field vary since some of the early cottrells were installed in existing flues and were designed to fit local conditions. The plate height has varied from 6' to 12' with 12' plates made from 16 gauge sheets being the current standard.

We have experimented with aluminum plates and wires in at least two installations and found little difference in the life of aluminum compared to the steel alloy. Economics ruled out the use of aluminum since the cost of aluminum plates was several times the cost of iron plates.

A few of the parameters for a single electrical field are given in Table II.

Table II
Parameters of a Basic Electrical Field

Item	English	Metric
Collecting electrode area	2604 ft.	242 m^2
Number of gas passages	15	15
Collection electrode spacing	6 in.	0.15 m
Gas passage length (active)	7.5 ft.	2.3 m
Discharge electrode spacing (along row)	3 in.	0.08 m
Discharge electrode diameter	0.08 in.	2.03 mm
Electrical volume (active)	651 $ft.^3$	18.44 m^3
Field length (active)	7.5 ft.	2.3 m
Field width (active)	7.5 ft.	2.3 m
Field height (active)	12 ft.	3.6 m

The active electrical volume in Table II is less than the total indicated by the field dimensions because there are four voids in each field. The lower frame of the discharge electrode assembly is supported by beams attached to the upper support and these beams are inside the field. Spacing between the grounded electrodes and the emitting electrode bottom support beams is not considered part of the active electrical field.

Our cottrells have been built with as few as two fields in series and with as many as six. However, the most common design is four fields in series. The number of parallel rows of fields is determined by the expected gas volume and our experience in operating cottrells on a similar gas stream. In some of our smelters we have either replaced a cottrell or expanded the existing cottrell. In our recently designed cottrells, we generally have had the measured efficiency for a similar type of operation and have used these data to size the new cottrell.

We normally design a cottrell on the basis of retention time in the electrical field. Cottrells built since 1974 have ranged from 7.9 seconds to 16 seconds retention time. These retention times result in a gas velocity of 3.8 (1.16 m/sec) and 1.9 feet per second (0.58 m/sec) respectively in a four field series. The cottrell designed for a 16 second retention time was installed on a copper converter gas stream. Our experience has indicated that particulate from a converter is the most difficult to collect and the combined particulate from copper roasters and reverberatory furnaces is one of the easiest to collect.

A basic electrical field contains fifteen rows of 26 discharge electrodes suspended at three inch intervals from a three inch diameter pipe. Each electrode is tensioned by a twelve pound "U" shaped iron weight attached to the free end. Beneath each row of wires, a 2-1/2" x 5/8" bar is so installed that the tensioning weights are suspended over it. The bar arrangement is such that when an electrode breaks the "U" shaped weight is retained on the bar thus preventing it from falling into the hopper and conveyors which are underneath.

The electrode suspension frame is supported by "I" beams placed at either end. The "I" beams are, in turn, supported by four stacks of insulating brick. Nominally, each stack is made up by alternating 7 and 9 inch square bricks, each 1-3/4" thick, for a total of eight bricks. Each stack of bricks is housed in a metal box located in the gas atmosphere below floor level. Since the supported "I" beam passes through an opening in this box the supporting bricks are exposed to the gas and particulate. Because of this there is a propensity for current to track or short across the stack of bricks. Cleaning the brick at weekly intervals reduces the frequency of shorting.

Each electrical field has a multiple number of cover plates to allow easy access for replacing broken electrodes and for major repair work. Our latest designed covers have an elliptical shaped gasket fastened to the bottom edges of each cover to exclude air infiltration. The older design covers do not have gaskets and the floor is generally covered with sheets of either plastic or rubber to reduce air infiltration.

Dust recovered in our cottrells is removed by either screw conveyors or drag conveyors, with screw conveyors the usual method for cold cottrells and drag conveyors for hot cottrells.

One plant has a combination screw conveyor system and air conveying system to return dust to the smelting circuit. In three of our cottrells dust is returned directly to the smelting circuit in closed conveying systems. Dust collected in other cottrells is put through a dedusting device prior to returning it to the charge preparation system.

Electrical

Our early cottrell electrical power supply included either a 15 or 25 KVA transformer for each field, a mechanical rectifier, and a hand-controlled power selector operating through a series of resistors for voltage control.

Over the years, starting in the 1950's and during the 1960's, the mechanical rectifiers were gradually replaced, first with selenium conversion kits, and then eventually with silicon conversion kits connected to the original transformers.

Starting in 1963, we experimented with an automatic voltage control device which adjusted the primary voltage through the use of a multiple number of mercury switches connected to existing resistor banks. As the state of the art advanced, we replaced the original automatic voltage regulators with saturable core reactors. As new cottrells were built, we installed saturable core reactor controls. Subsequently, thyristor controls have replaced saturable core controls on two of our converter cottrells. Our most recently commissioned cottrells have all been equipped with thyristor controls. The power supply for these cottrells consist of 45 KV-1000 MA transformers with built-in rectifiers and thyristor automatic voltage regulators.

We found by experimenting on a copper converter cottrell that thyristor controls gave a substantially higher power input than was possible with saturable core reactor controls on this variable gas stream. Some of our plants which have a constant gas condition, and in which saturable core controls were originally installed, continue to operate with this type of control.

During the transition from mechanical rectifiers to the present automatic controls, the number of electrical fields connected to each transformer was changed. We currently have 3, 4, or 5 fields per transformer on our own designed cottrells depending on the cottrell configuration. We have one cottrell, of a commercial design, which has a power supply for each field.

In the ASARCO plan, the overhead power distribution system is arranged so that any transformer can be connected to any electrical field in the cottrell. The normal arrangement is four fields connected to each transformer with no two fields in series on the same transformer. Under these conditions, a short in one field still leaves three fields operating in each of our adjacent sections. When one transformer becomes inoperative, the four connected fields are individually connected to four other transformers. These four transformers are then temporarily operated with five connected fields.

Our reason for connecting a minimum of three fields per transformer is dictated by the method used in cleaning the wires and plates in our cottrell. We clean by interrupting gas flow with dampers and disconnecting power to each of the fields in series with pneumatically operated switches. Disconnecting all units on the transformer rectifier (TR) sets could result in damage to the electrical equipment during a cleaning cycle if only two units were connected to a TR set and one of these units was previously disconnected for maintenance.

Vibrators

The early cottrells installed in our company employed specially-designed Chicago Pneumatic or Sullivan Company air vibrators to clean the plates and wires. These original vibrators were mounted above the floor with normally four vibrators being mounted on the plate frame and one on the wire frame. The wire frame vibrator incorporated a spring in its design to retract the pounding bar to prevent shorting after the cleaning cycle was complete.

The cost of the originally designed vibrators became too expensive since they were no longer a standard production item. Consequently, in 1968 we built our first cottrell with standard vibrators of a different design mounted under the floor and bolted directly to the support frames. In this instance, two vibrators were installed on the grounded frame and one on the wire frame. Air to the vibrators was piped through the floor with flexible tubing. This first cottrell with under the floor vibrators operate at 250-300°F (121-149°C) and the vibrators function quite well.

Our next cottrell was installed on a 600°F (315°C) gas stream. Due to the elevated temperature the vibrators had to be mounted above the floor. The discharge electrode frame vibrators were mounted on tapered ceramic shafts to provide electrical insulation. A small box was placed above the floor around each ceramic shaft and a thin plastic membrane was installed across the bottom of the box and around the shaft to provide a gas free environment for the upper portion of the shaft. The need for providing a gas free environment was dictated by the high acid dew point of the gas stream which could result in acid condensing on the ceramic shaft, thus causing current tracking to ground through the conductive film of condensed acid.

Maintenance

Wire electrodes are the major source of daily routine maintenance in many of our cottrells. Two of the cottrells which use both acid and water for conditioning experience 2 to 4 broken wires per day, as an average. In other cottrells, 30 to 50 broken wires per year are a common occurrence. Many of the broken wires are caused by arcing, particularly in those cottrells which use either no conditioning or water only for conditioning. At plants that have acid conditioning, corrosion is a major cause for wire failures.

A major overhaul of an electrical field is quite often performed by plant personnel with the crew consisting of an experienced cottrell operator, plus two laborers. One electrical field can be completely rebuilt in ten operating shifts with this three man crew. A complete overhaul consists of removing all the wire electrodes, the electrode support frame, and all the plates. Individual corrugated plates are either 24" or 31" wide and 12' long and are light enough for one man to handle although two men generally are needed in lining up the plates during installation. The only shop personnel needed is a welder to tack weld the sheets to the top support.

The frequency of a major overhaul in our smelter cottrells is quite variable ranging from four years to over twenty years, depending primarily on acid condition and operating temperature of each cottrell.

The cottrells which discharge to a stack can undergo major repairs with no upset in the operation. During an overhaul of one electrical field, the practice is to close both inlet and outlet dampers in that section and remove all covers over the field to give an unobstructed access to plates and wires. Repair work is generally conducted on day shift only. In some instances, temporary covers are installed on the unit under repair so the other three fields can be placed back in operation during afternoon and night shift. Those cottrells which are on chemical plant gas streams cannot be completely overhauled unless the chemical plant is down since air leakage past the damper dilutes the SO_2 content of the gas stream. In the context of this paper, the term chemical plants refers to our liquid SO_2 plant or to our seven sulfuric acid plants.

Labor

Labor required to operate our cottrells range from two men per shift, plus a dust puller on day shift, to some cottrells on chemical plants where the chemical plant operators are assigned the cottrells as a part-time job along with their regular job.

There are eight chemical plants in the company, six chemical plant cottrells have men who monitor the cottrell as part of their other duties, one plant has two men on day shift only, and one plant has operators on all three shifts. Two cottrells on reverberatory or roaster-reverberatory gas have two men on each shift because both acid and water are used for conditioning, while a third reverberatory-roaster cottrell operates with two full-time operators on day shift since water only is used for conditioning and water conditioning is automated in all our cottrells.

The operator's job varies to some degree among plants; but generally, they include removing shorts, cleaning insulator boxes and applying hydrated lime on the support insulators, taking power readings, cleaning water sprays since acid plant scrubber water is used for cooling gas ahead of the cottrells, adjusting the acid addition, adding hydrated lime to the gas stream for conditioning, etc. Two of our cottrells do not have an automatic cleaning system and for these two units, the operators manually close dampers, disconnect power, and activate air hammers.

Operation

All of our cottrells are monitored by either cottrell operators, chemical plant operators, or by supervisory personnel inspecting the cottrell at frequent intervals.

A broken wire electrode is indicated by erratic fluctuations in the voltmeter and ammeter connected to the transformer which supplies power to the field which has a short. These meter anomalies can be in the form of wide swings in the readings, as is the case of an oscillating wire, or very low but steady voltmeter reading if there is a dead short. Most of our older cottrells have, in addition to meters, a light connected to the primary circuit, which glows bright during high power input, dims with a voltage decrease,and flickers when there is a fluctuating short which usually indicates a broken wire. This light arrangement gives the operator a visual indication of applied power and is used in locating which field has a short. When a broken wire is indicated in one of the fields served by a specific transformer, the operator disconnects power on each field one at a time until the faulty field, as indicated by a return to normal of the light or meter, is located.

In those cottrells discharging directly to stack, the inlet and outlet dampers are then closed, one cover removed near the area of arcing, the safety bar lifted, power reapplied, and the specific point of arcing noted. Broken wires can be located very rapidly with this method and can be replaced with a new wire in a matter of some fifteen minutes. In those cottrells which are part of a chemical plant gas cleaning system, broken wires are located and removed as described above but generally are not replaced at that time. Air leakage through the dampers dilute the gas stream so the time covers are open is kept to a minimum.

A low voltage and higher than normal steady milliampere reading generally means that current is tracking to ground through one of the four stacks of insulator bricks supporting the discharge electrode frame. When this occurs the operators replace the stack of insulating tile which is tracking to ground.

As regards to power-on cleaning versus power-off cleaning, we have experimented with keeping power on the last field during the cleaning cycles to recover dust drifting from the other fields during a cleaning cycle. The last field was cleaned only once each shift during these tests. EPA Method 5 sampling did not indicate any significant difference in grain loading with and without power on the last field during the cleaning cycle.

Dust in our precipitator is normally allowed to accumulate over night in the hoppers and is removed on day shift. We seldom have trouble with accumulated dust catching on fire. However, this has happened once at our East Helena lead smelter when dust inadvertently accumulated and spontaneous combustion ignited the dust causing it to fuse into clinkers. Spontaneous combustion in reverberatory and roaster cottrells is a very rare occurrence and has never happened to the extent of damaging any of the conveying systems or the cottrells.

Several of our cottrells are designed with the inlet distribution plenum at right angles to the cottrell sections. This arrangement is generally dictated by the plant configuration, and as a result of these poor inlet conditions, we balance gas flow through the cottrell by adjusting either the inlet or the outlet dampers. This system is monitored by the use of a manometer, connected to the first and last field in each section, which measures the pressure drop across each section. If the manometer deflection on all sections is not uniform, dampers are biased toward the closed position on those sections exhibiting the higher manometer deflection.

All but two of our own designed cottrells have an automatic cleaning cycle which is activated at either one hour or two-hour intervals, depending on the type of operation. The sequence of actions during a cleaning cycle are: close inlet and outlet dampers, disconnect power on each field with a pneumatically operated switch, activate air hammers for 30 to 60 seconds, reconnect power, and open the dampers. Once the cleaning cycle starts, all sections are cleaned one at a time.

All three of our cottrells on reverberatory or roaster-reverberatory gas streams which discharge direct to stack are cooled to 225-300°F (107-149°C) with water sprays. These cottrells are operated at lower temperatures to condense volatile metal oxides and to increase recovery of these compounds. Our four cottrells on copper converter gas are part of chemical plant gas cleaning systems, three of which cool the gas to 280-300°F (138-149°C) while the fourth cools the gas to 400°F (204°C). Cottrells on our fluid bed zinc roasters operate at 600-700°F (315-370°C). The only cottrell operating on an updraft lead sinter machine at the time this paper is being written operates at 400-450°F (204-232°C). Dry cottrells installed on the zinc and lead sintering operations are part of chemical plant gas cleaning systems. All our chemical plants have a second stage gas cleaning system following the dry cottrell and volatile metals escaping the dry cottrell are recovered in this system. The second stage systems consist of scrubbers and wet cottrells.

Particulate Parameters

Particulate loading into our cottrells is quite variable depending on the particular operation. The measured inlet grain loading for one cottrell, cleaning a combination reverberatory and roaster gas, is 2.5 gr./SCF (5.7 gm/m^3). Our Hayden copper converter cottrell has a measured grain loading of 2.0 gr./SCF (4.6 gm/m^3). The calculated grain loading, based on tons of dust collected and normal gas flow rate, for the lead sinter plant cottrell is 1.5 gr./SCF (3.4 gm/m^3). For one of our zinc plant cottrells the calculated grain loading is 0.6 gr./SCF (1.4 gm/m^3).

A series of tests for particle size distribution for a roaster-reverberatory cottrell and for a copper converter cottrell were conducted using a Mark III impact sampler with a preimpactor and using Reeve Angel substrates which were not conditioned prior to the sampling. Results of these tests are shown on Figure I.

A series of tests were conducted on the outlet of the Hayden roaster-reverberatory cottrell for particle size distribution and the results are shown on Figure II. These particle size samples were collected at a single point approximately 30' downstream from the cottrell outlet.

Data collected from traverse sampling of the Hayden roaster-reverberatory cottrell for grain loading, and the particle size distribution data were used to calculate the cottrell efficiency for various size fractions and these data are shown on Figure III.

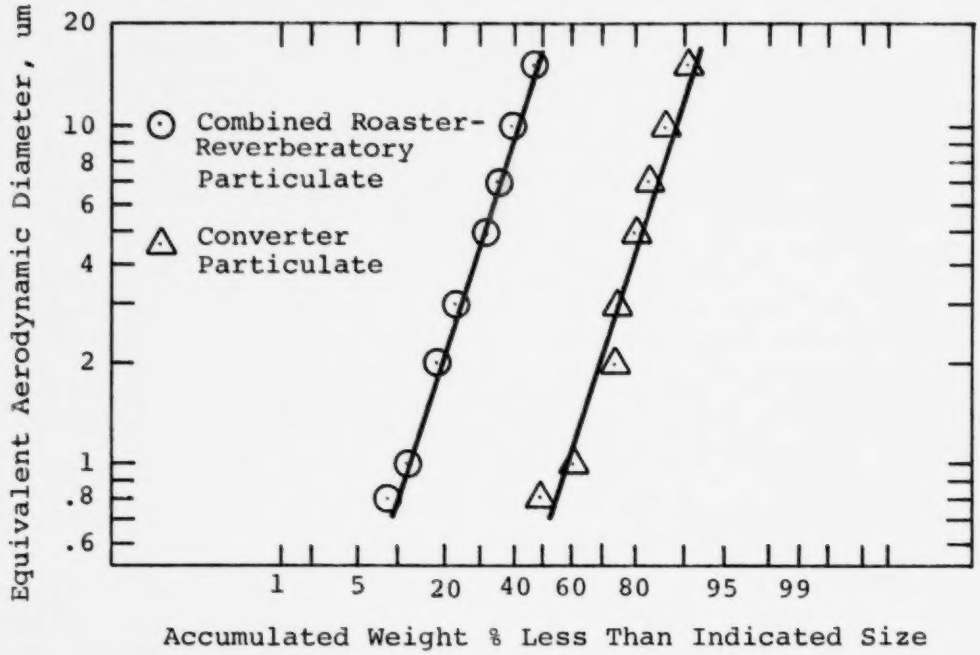


Fig. I - Particle size distribution at the inlet of two cottrells.

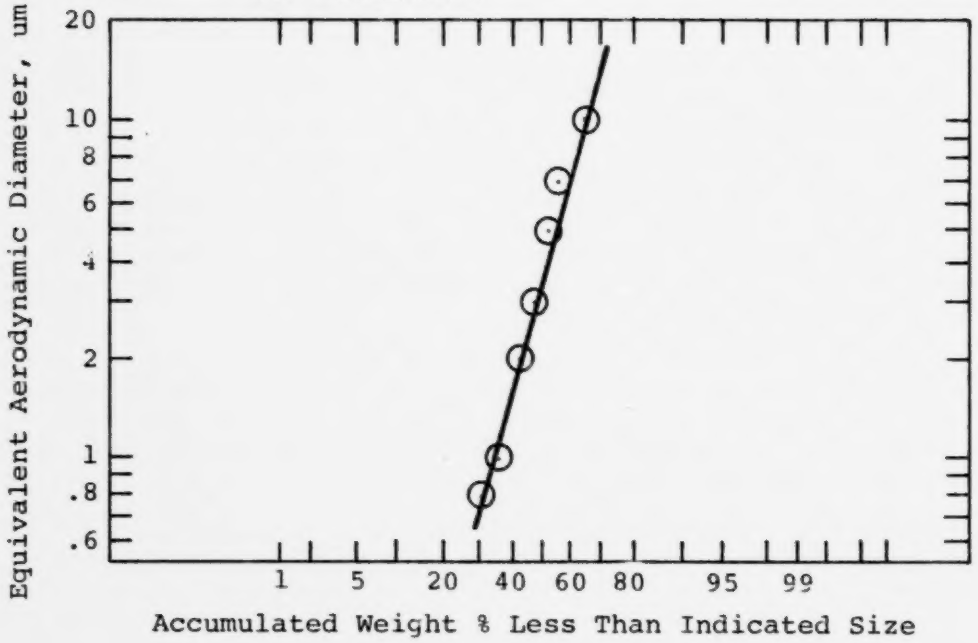
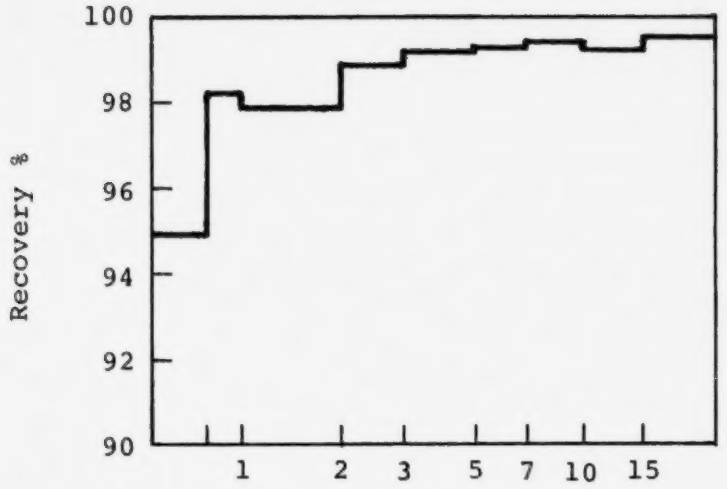


Fig. II - Particle size distribution for a roaster-reverberatory cottrell outlet.



Equivalent Aerodynamic Diameter, µm

Figure III - Roaster-reverberatory cottrell efficiency with reverberatory gas only conditioned with water.

The particulate characteristics play a major role in determining the current flow in a cottrell. The figures below indicate the two extremes in our operations for resistivity of particulate collected and characteristics of the gas stream to conduct current. Particulate collected in our roaster-reverberatory cottrells contain copper, generally in the range of 15 to 24% of the total dust content, with 2 to 5% each of lead and zinc. The particle size measured at the inlet of our Hayden roaster-reverberatory cottrell is advantageous in efficient recovery with only 12% by weight being under one micron in size. Particulate collected in our copper converter cottrells contain lead, generally in the range of 30% of the total dust content, with 3 to 5% copper and 7 to 12% zinc. The particle size entering the Hayden copper converter cottrell has 60% by weight under one micron in size.

Data for a copper converter cottrell are given in Figure IV where current flow per square meter of plate surface is plotted for the three electrical fields. This cottrell has four parallel sections each of which is three electrical fields deep. Each field is connected to its own individual TR set with the inlet field in each section being designed as the first field of the cottrell. Gas to the cottrell is conditioned by the addition of water.

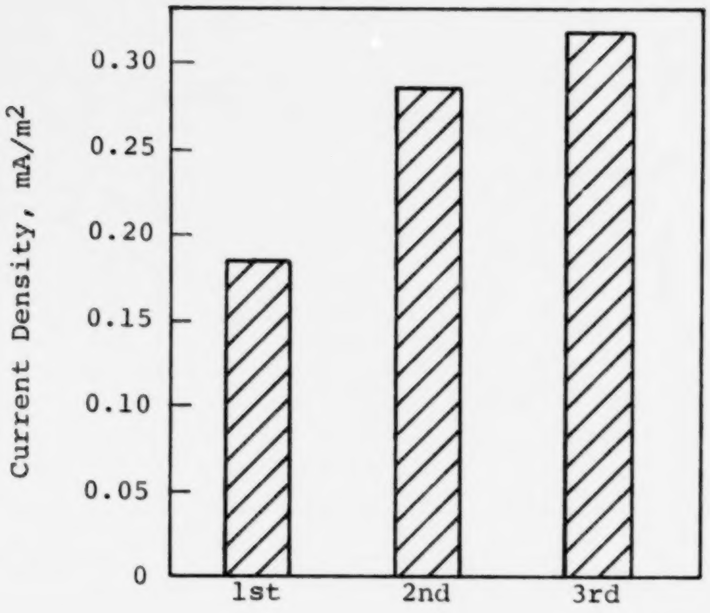


Fig. IV - Current flow in a copper converter cottrell.

Similar to Figure IV, Figure V presents a plot of current flow per square meter of plate area for a roaster-reverberatory cottrell in which the reverberatory gases are water conditioned but the roaster gases are unconditioned. This cottrell is an arrangement of eight parallel sections of four fields in series with the inlet field being considered the first field.

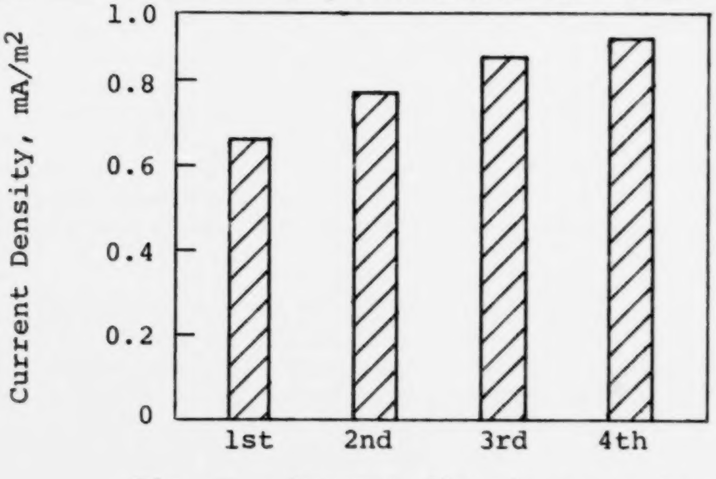


Fig. V - Current flow in a roaster-reverberatory cottrell.

Gas Conditioning

ASARCO first discovered the importance of conditioning shortly after starting a pipe cottrell on converter gas at our Garfield smelter in 1912.(4,5) It was discovered in the preliminary experiments that dust collected on the pipes were conductive during the finish blow due to the higher acid content in the gas stream. During the slag blow a non-conductive dust layer was formed and further precipitation ceased. It was found, however, that by injecting a small amount of moisture the dust layer was rendered conductive and the process worked satisfactorily at about 194°F (90°C).

Some of our early smelters had no need for conditioning, except for copper converters, due to the high sulfur content of ores being smelted at this time and with natural formation of SO_3 for conditioning. Later with a change in ore supply, with lower sulfur content, the natural SO_3 formation decreased and conditioning assumed an important role in our cottrell system design. Water conditioning to humidify the gas was one of the first steps in conditioning followed later by the use of sulfuric acid.

One of the first efforts in acid conditioning occurred in the 1920's at our Murray, Utah lead smelter where sulfuric acid was boiled in iron pots and the acid vapor plus combustion gases were inspirated into the flue system by flue draft.(6) Our Tacoma smelter was also using this method for acid conditioning during this same period. Boiling sulfuric acid limited the quantity of acid which could be evaporated and the second step in acid conditioning was through the use of a stoker-fired coal box with sulfuric acid sprayed into the products of combustion. This system allowed the use of up to 2-1/2 tons of acid per day. This method was later refined by the use of a small brick-lined furnace in which sulfuric acid was sprayed into the flame of a gas burner and then was inspirated into the flue. This procedure proved to be wasteful because a portion of the acid was broken down to SO_2. This situation, coupled with the fact that the furnace was a high-maintenance item, led to the discontinuance of the gas fired furnace.

Our next step in developing a system for acid conditioning, and one which we are currently using, was to atomize acid through sprays with high-pressure air. The acid is sprayed into a flue system ahead of any water conditioning and where the gas temperature is 300°F (149°C) or higher.

One of the primary deficiencies in the use of acid conditioning is the lack of a method for optimizing acid content of dust collected in the cottrell. We have experimentally tried several systems to control acid addition but have not been successful with any method tried and, as a result, all our cottrells using acid conditioning have two men around the clock to control acid addition along with other duties.

An experiment was conducted at one of our copper converter cottrells to determine the effectiveness of hand controlled acid addition using the milliamp meter and spark rate in the cottrell as a guide for acid conditioning control. Acid content of collected converter dust samples taken each hour were used as the criteria for judging optimum range. The empirical method used for acid determinations involved leaching a sample of dust in water, filtering the sample, and titrating to a methyl orange end point with sodium hydoxide. Acid content less the 0.5% is too low, between 0.5 to 1.5% is considered optimum, between 1.5 and 5.0% is acceptable, and over 5.0% is too high. An example of the difficulty in acid addition control is indicated by the results of this test. Acid control was good for 15.7% of the time, acceptable for 39.3% of the time, and either not enough or too much 45% of the time.

SO_3 conditioning was installed at one of the Tacoma cottrells when changes in the operating conditions resulted in gas too cold for effective evaporation of acid. The source of SO_3 was the copper converter acid plant where a small side stream of gas was taken from the acid plant converter and introduced into the flue system. This resulted in an improvement in cottrell effectiveness over that possible with acid addition. However, we were faced with problems similar to those faced when using acid, no method for automatic regulation of SO_3 addition.

Lime conditioning is used at two of our converter cottrells during the copper blow when the natural formation of SO_3 is highest. The segment begins at the first part of the copper blow and continues through the rest of the cycle. Formation of SO_3, with little or no production of particulate, results in wet plates and wires in the cottrell when operating in the 250-300°F (121-149°C) temperature range. This condition results in a drop in voltage and an increase in milliamps. Such an electrical change is an indication to the operators that acid neutralization is required. Achievement of control is realized by blowing hydrated lime into the gas stream at a point upstream from the cottrell. The conditioning continues until the acid is neutralized to a point where the voltage and milliamp readings return to normal range.

The cottrell installed on our Hayden copper converter gas stream ahead of the acid plant is not ASARCO's design and was installed with no provisions for conditioning the gas stream. The intent was to operate at 750°F (400°C) to control resistivity of the dust in an acceptable range. Temperature control was seldom uniform enough for good resistivity control due to variations in converter activity and the designed cottrell efficiency was seldom, if ever, realized. The efficiency was not acceptable and a cooling chamber was installed ahead of the cottrell where both water and acid could be used to condition the gas.

Saturable core reactor controls which were installed as original equipment on this cottrell were found to react too slowly to rapid changes in gas conditions as converters were turned in and out of the hoods. A thyristor control was installed on one inlet field to compare the performance of these two different types of controls. The thyristor controlled field and other inlet fields with saturable reactors had essentially the same power input. The two fields, which were still saturable core controlled, following the thyristor control indicated 49 to 68% increase in applied power over similar fields with the original control on the inlet field. This indicated to us that the thyristor control resulted in a lower particulate loading than the saturable core reactor could achieve. As a result of this test, thyristor controls were installed on all twelve fields.

Several emission tests were conducted on the outlet of one of the four sections using a ceramic in-stack filter. These tests cover the period before and after thyristor controls were installed and with and without water conditioning. Results of these tests are given in Table III.

Table III
Mass Emission Data - Copper Converter Cottrell

Item	Conditioned Saturable Reactor	Conditioned Thyristor	Unconditioned Saturable Reactor	Unconditioned Thyristor
Gas Temperature °C	183	180	217	227
Water Content % by Volume	6.1	3.5	1.6	1.3
Current Density Milliamps/m^2	0.33	0.45	0.16	0.32
Current Density Milliamps/1000 ft.^{2}	30.6	41.6	14.5	29.6
SO3 Concentration PPM		209		600
Particulate Concentration Gr./DSCF	0.0546	0.0226	0.271	0.0545
Particulate Concentration mg/DSCM	125	52	621	125

Due to its high resistivity particulate from copper converters is the most difficult dust to recover of any we have in our smelters. This condition is indicated by lower current density than we have on other operations shown in Table IV.

Table IV
Typical Cottrell Operating Parameters

Item	Converter	Reverber-atory Roaster	Lead Sintering	Zinc Roaster
Gas Velocity FPS	1.5	11	3.2	2
Time in Electrical Field Seconds	20	2.7	9.4	11.2
Gas Temperature °C	140	121-149	200	300
Water Content % by Volume	8.0	9.0	7.5	N.A.
Current Density Milliamps/ m^2	0.28	0.52-0.84	0.7	0.85
Current Density Milliamps/ 1000 $ft.^2$	26	48-78	64	79
Applied Voltage KV	24	28-31	25	27

Copper converters operate as a batch process in which during the first part of a cycle, the slag blow, most of the lead and zinc is fumed out of the matte. During this part of the cycle some of our smelters add acid to condition the dust even though the gas stream contains SO_3. A series of samples were taken at one of our copper plants to determine the SO_3 concentration during the slag blow and during the finish blow. During the slag blow, the SO_3 concentration ranged from 59 to 315 ppm with 156 ppm as an average for the nine samples taken. Power to a cottrell is lower during this phase, due to the dust burden, than it is during the second phase of the cycle called a finish blow. During a finish blow there is relatively little particulate formed and the SO_3 content of the gas stream average some 1240 ppm. The range of SO_3 concentration for sixteen samples collected during a finish blow when no water was used to quench the gas temperature was from 600 to 2654 ppm. The ideal situation would be to blow one converter on slag, another on finish, and combine the two gas streams but it is not always possible to keep this combination of one on finish and one or more converters on a slag blow.

There are two principle reasons for lower SO_3 during a slag blow than during a copper blow in our operations. Temperature plays a major role in the conversion of SO_2 to SO_3. Temperatures measured during these tests ranged from 800°F to 950°F (427-510°C) during a slag blow and from 800°F to 1200°F (427-649°C) during a copper blow. The higher temperature during a copper blow increases the SO_2 to SO_3 conversion rate. The second reason for greater SO_3 formation during a copper blow is the higher concentration of SO_2 during this phase of the operation.

When acid plants were first installed on converter gas and it became necessary to reduce air infiltration into the converter hoods, the increased hood gas temperature and higher grade SO_2 in the gas stream promoted the formation of SO_3. This resulted in a wet, sticky dust in the cottrell. One of our first attempts to overcome the problem of high acid dust collected was to blow hydrated lime into the converter hood; up to five tons per day at one plant. The cost of lime, and maintenance of the lime feeder, were factors which made this method of controlling SO_3 concentration undesirable. The problem was alleviated, but not completely solved, by spray cooling the converter gas to 800°F (427°C) at the hood.

We ran a series of tests at one of our copper plants to determine the effectiveness of water cooling to control SO_3 formation during a slag blow with and without water cooling due to the normal lower gas temperatures during this phase. During a copper blow the pounds of SO_3 were reduced by 50% with water cooling the gas to 800°F (427°C). Figure VI is a plot of converter gas temperature versus Kg. per minute of SO_3 measured when the converter was on a finish blow.

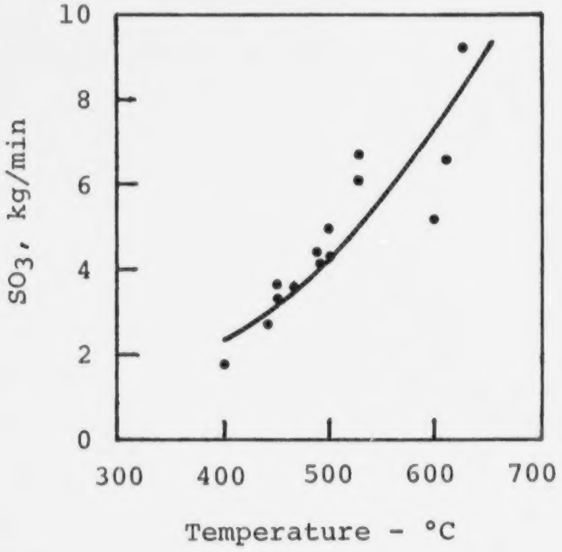


Fig. VI - SO_3 formation in a copper converter gas stream during a finish blow. Gas volume 40,500 scfm.

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ELECTROSTATIC PRECIPITATORS (ESPS) IN TWO STAGES USED FOR ARSENIC RECOVERY AT THE RÖNNSKÄR COPPER WORKS

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ABSTRACT

A modern dedusting plant was placed in operation in 1975 at the Rönnskär Copper Works in northern Sweden for the treatment of gas from the roasters. A selective recovery of dust has been achieved through the use of electrostatic precipitators (ESPs) in two stages. In the first stage copper, zinc and lead are collected at temperatures exceeding 300°C. The solid arsenic is collected in the second stage after the gas has been cooled to 120°C. The remaining impurities including gaseous components are washed in a wet stage before the gas is transported to the sulphuric acid plant.

This paper reviews the background - a prestudy phase, including pilot-testing - design of the full-scale plant, and operating experiences.

BACKGROUND

The Process

The Rönnskär Copper Works treats sulphidic concentrates with complex composition. Half of the concentrates come from Boliden's own mines in Sweden. The other part is purchased. The concentrates contain approximately 20 - 25% copper and 1% arsenic. Lead, zinc, nickel, arsenic, selenium, sulphur and precious metals are recovered besides copper. Roasting, the first step in the copper process is done in four multi-hearth furnaces.

The Old Plant

Previously, the gas from these furnaces was cooled to a temperature of about 150°C and then treated in four old ESPs for particulate collection.

The dust collected in the coolers and the ESPs contained about 9% copper, 4% zinc, 5% lead, 20% arsenic and 0.6% mercury. Because of its high content of arsenic and mercury this dust could not be completely recycled to the process. Consequently about 1500 tonnes of collected dust had to be disposed of every year, resulting in a large wastage of useful metals.

Due to extensive corrosion in the old installation, high volumes of air in-leakage diluted the process gases. This increasing treatment costs in the sulphuric acid and liquid sulphur dioxide plants.

The New Concept

In 1972 Boliden and AB Svenska Fläktfabriken undertook a joint research and development project aimed at solving these problems. The objective was to separate the dust into two fractions, one containing copper, lead and zinc, the other mainly arsenic and mercury. The first fraction would be recycled to the smelter, the second would be treated for recovery of arsenic and mercury.

Most arsenic and mercury compounds gasify at a temperature of 300°C while the copper, zinc and lead compounds remain in a solid phase. Filtration of the gases at or above this temperature in the first filtering stage, would keep the recovered dust relatively free from arsenic and mercury and allow for recycling. The gases free from copper, zinc and lead could then be cooled to solidify the arsenic and mercury which can then be removed in the second filtering stage. Fabric filters were judged unsuitable for this service because the gas is rich in SO_2 (approx. 5%) and there is a risk of condensation. ESPs were suggested for both high- and low- temperature service in the two stages of filtration.

A prestudy, including pilot plant tests, was undertaken in order to investigate the operating conditions for the two ESPs.

PRESTUDY

Prevention of Glass Arsenic Formation

One point of concern when cooling gases containing As is the formation of so called glass arsenic. This is a very hard sintered compound which could be formed during condensation of vaporized arsenic in the temperature range of 175 - 250°C. Glass arsenic cannot be easily dislodged by conventional rappers and vibrators and frequent manual clean-up is required. Furthermore dust transportation systems can be damaged or blocked by hard pieces of glass arsenic. Temperatures within the above range must therefore be carefully avoided for all materials that come into contact with the gas. This critical gas temperature range should also be passed as quickly as possible which could be achieved by using an evaporative cooler, where atomized water is sprayed into the gas stream.

Possible Resistivity Problems with As_2O_3

At the time the electrostatic precipitation properties of As_2O_3 were largely unknown. The operating temperature of an ESP used for this should not exceed 130°C to avoid loss of arsenic in vapour form. According to the International Critical Tables the amount of As_2O_3 in vapour form at 130°C can be in the order of 60 mg/Nm^3.

The resistivity of As_2O_3 as measured in the laboratory at 130°C and a water dew point of 40°C was found to be as high as 10^{15} ohm cm. ESP operation with such a resistivity would certainly create severe back corona on the collecting plates, resulting in arcing and/or reentrainment of the dust. The above effects would reduce the efficiency of the ESPs.

The upper limit of resistivity of dust for efficient precipitation is normally in the order of magnitude 10^{12} - 10^{13} ohm cm, depending on the application, as well as the electrode geometry. The extent of back corona would be minimized by using discharge electrodes which give an even current distribution over the entire collecting surfaces. This is discussed in detail in (1).

The dust resistivity can be reduced, for example, by increasing the water content of the gas (2). The necessary humidification would be provided by the evaporative cooler. Due to the high SO_3 content of the gas - resulting in a high acid dew point - a reduction of the resistivity in the actual gas was also to be expected.

Pilot Plant Testing

Two major items were to be investigated:

1. Extent of As_2O_3 separated in a hot-side ESP.
2. Conditions under which As_2O_3 can be precipitated.

A two field pilot ESP was used for the testing. Each field had a height of 2 m, depth of 2.4 m and a width of 0.25 m. The temperature drop across the ESP was minimized by using external heating of the walls. The gas to be tested was sampled at high temperature from the gas duct after one of the roasters. The desired operating gas temperature was obtained by cooling the gas upstream the ESP.

Different dust and gas conditions were tested by varying the type of concentrate in the roasters, table 1.

Table 1. RESULTS OF TESTS WITH THE PILOT ESP

Test series	As in conc. %	T oC	A/Q m2/m3/s	Efficiency particulate %	S_{in} g/Nm^3 total	S_{in} g/Nm^3 As_2O_3	As_2O_3 total g/Nm^3
A	0.4	300	40	85	4.5	0.4	0.8
B	5.5	300	70	95	10	1.0	20
C	12	130	70	98	24*	23*	50*

* The As_2O_3 and dust concentrations were measured at around 180°C.

T = Mean temperature in the pilot ESP

A/Q = Specific collecting area

S_{in} = Inlet dust concentration (particulates) to the ESP

As_2O_3tot = Total As content (as As_2O_3) in flue gas before the ESP

Dust concentrations were measured using an in-the-duct-type sampler with glass fibre thimbles. The sampled gas was then passed through impinger bottles at ambient temperature where condensables were collected.

A very high As content in the concentrate was used during test series C to achieve conditions in the pilot plant ESP to resemble those in the cold-side ESP in the full scale plant. No hot-side collector was used during this test series. The series C resulted in small amounts of metal "impurities" in the collected dust, table 2.

Table 2. ANALYSIS OF DUST COLLECTED IN PILOT ESP, IN %

Test series	A	B	C
(As in conc.)	(0.4)	(5.5)	(12)
As	6 - 12	14 - 20	66 - 71
Cu	10 - 12	15 - 20	0.5 - 1.7
Pb	17 - 21	6 - 11	0.4 - 1.1
Fe	4 - 7	6 - 12	0.5 - 2.4
Zn	5 - 6	3 - 4	0.2 - 0.5
Hg	0.062	0.001- 0.03	0.02 - 0.04

The amount of As collected in the hot-side pilot ESP compared to total As in flue gas as a function of temperature is shown in figure 1.

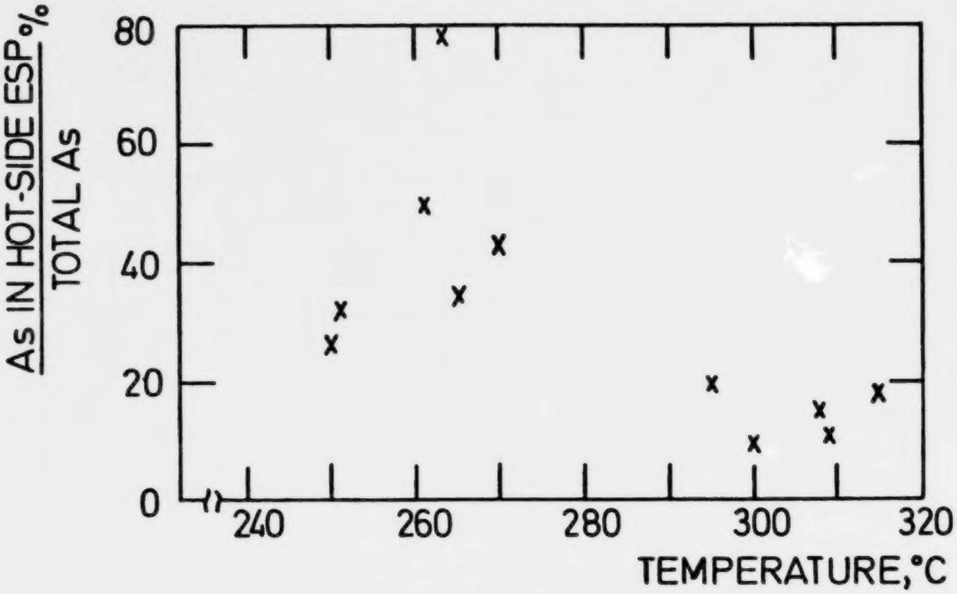

Figure 1. Influence of gas temperature on percentage of As collected in the pilot ESP

A minimum gas temperature of 300°C should be maintained in the hot-side ESP to reduce the As_2O_3 content of the collected dust recycled to the smelter. On the other hand, as the dust might be of a pyrophoric nature, the temperature in the hot-side ESP should not be higher than necessary. The operating temperature must therefore be very carefully controlled.

FULL- SCALE PLANT

Layout in Principle

Figure 2 gives an overall view of the ESP stages. Cyclones placed close to the outlet of each furnace collect the coarse dust in the off gases. This dust is returned to the furnaces. The gases from the four furnaces are then mixed together in a mixing chamber before entering the ESPs in order to equalize variations in the gas temperature and gas composition. The outlet gases from the mixing chamber are fed to two parallel gas circuits each comprising one hot-side ESP, one cooling tower and one cold-side ESP in series. These are followed by one single wet stage, where the gas is washed in a tower and cooled indirectly in a cooler with cold sea water before the gas can be utilized for production of sulphuric acid and liquid SO_2, figure 3. The used water containing As as arsenite is circulated back to the cooling tower between the ESPs.

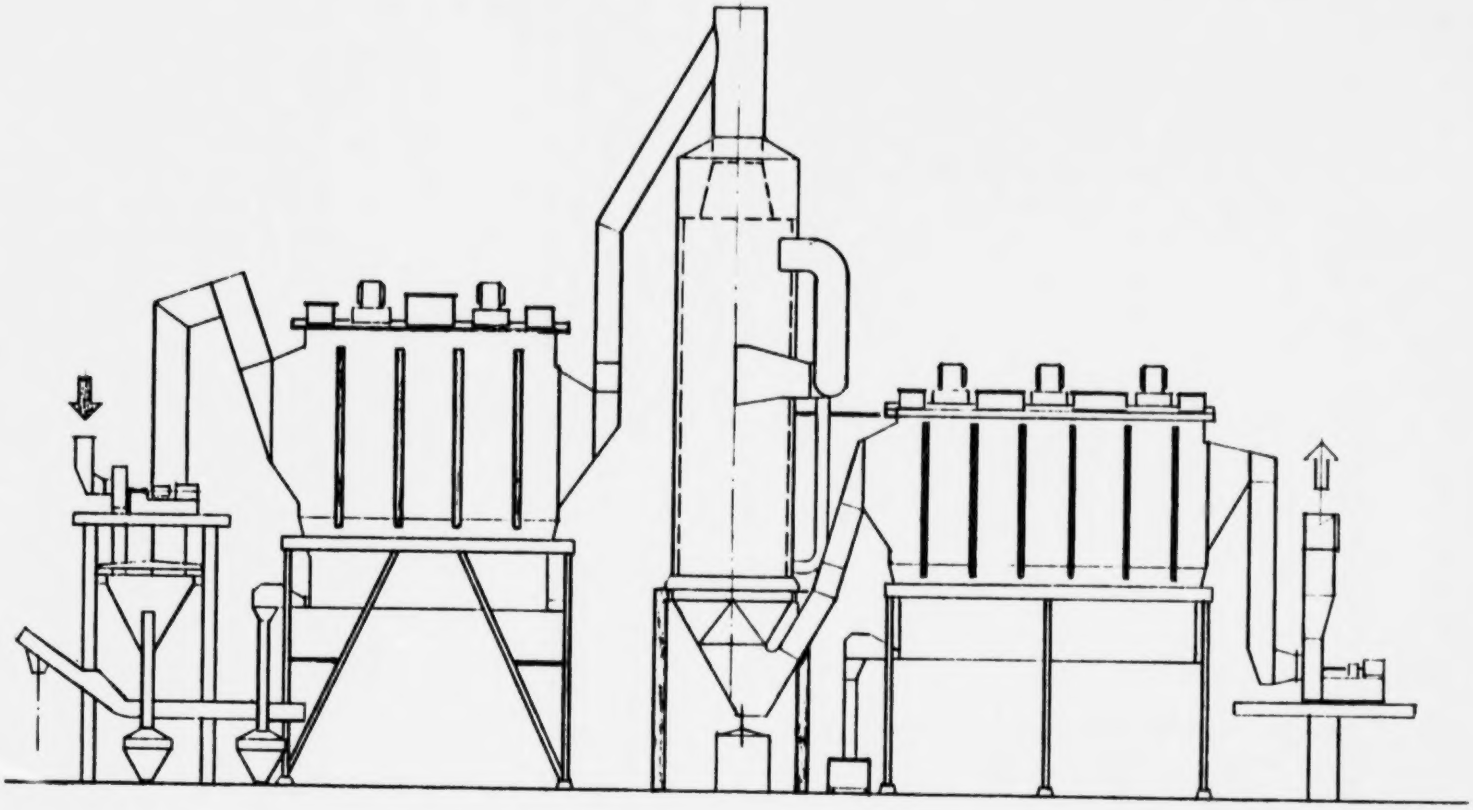

Figure 2. Overall view of the selective dust cleaning at Rönnskär Copper Works

Surplus water is cleaned by using sodium sulphide in the central water cleaning plant of the Rönnskär Works.

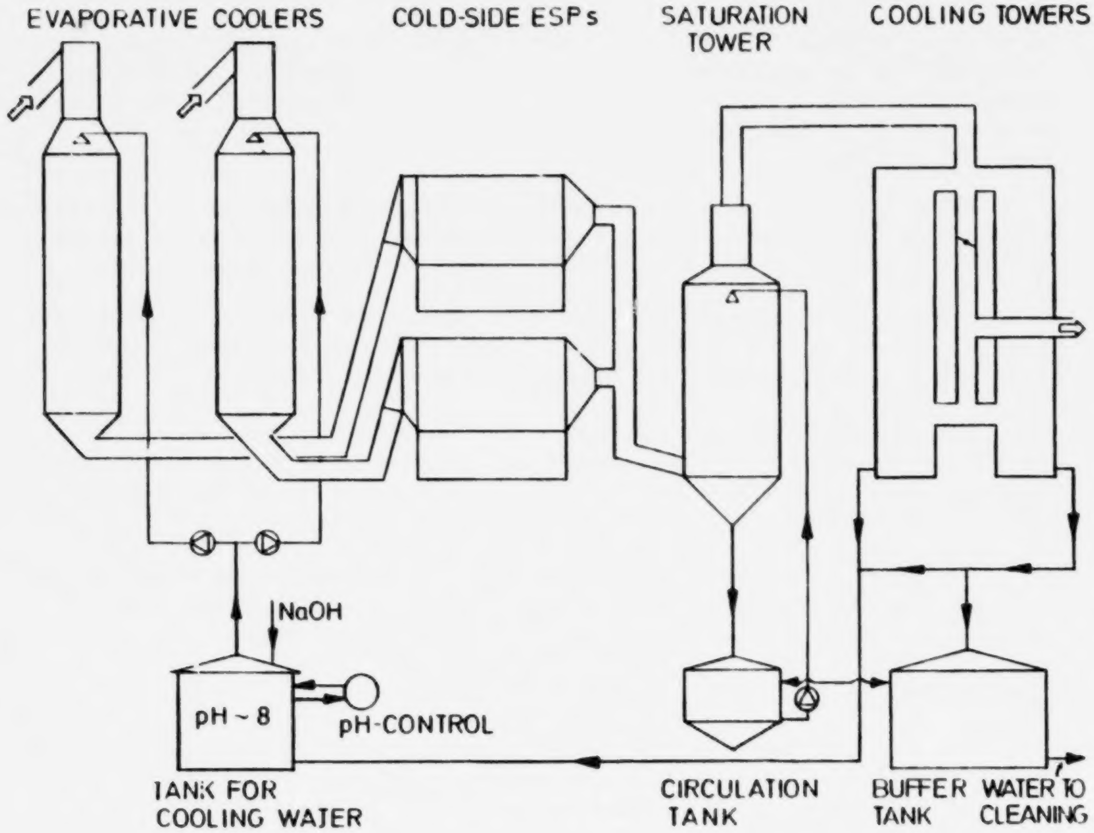
EVAPORATIVE COOLERS
COLD-SIDE ESPs
SATURATION TOWER
COOLING TOWERS
NaOH
pH ~ 8
pH-CONTROL
TANK FOR COOLING WATER
CIRCULATION TANK
BUFFER TANK
WATER TO CLEANING

Fig 3. Sketch of the wet stage and the water flows

Design of the ESPs

The ESPs are of heavy-duty type for industrial applications. The gas flow is horizontal. The collecting plates hang from roof beams, forming gas passages in which the high voltage frames are located. Each frame is supported by four insulators. The discharge electrodes are of the spiralized wire type and are hooked on to the frames. The spiral shape provides a number of evenly distributed points from which corona takes place, and the design is well-proven in high-resistivity applications. Owing to the properties of a wire the micro- and the macro- current distributions are excellent. Each bus section is energized by a thyristor controlled transformer rectifier set equipped with automatic spark rate controllers.

Rapping is achieved by tumbling hammers mounted on rotating shafts. An alarm device signals if a shaft stops rotating.

The ESPs are equipped with trough hoppers; the hopper walls are electrically heated to a controlled temperature in order to avoid condensation and prevent formation of glass arsenic. Dust storage time in the hot-side ESPs is minimized due to the pyrophoric character of the dust. Redler conveyors operating continuously quickly remove the collected dust which is further transported by a drag-chain convyeyor to the electric smelter.

The collected dust in the cold-side ESPs is transported to the arsenic refinery in an enclosed system comprising redler conveyors and a vacuum operated pipe line. The dust is then processed to pure arsenic oxide.

A flushing system protects the ESP insulators from condensation and dust build-up. Air is heated to 300°C and 150°C respectively in the two stages and is fed through the insulators into the ESPs.

The hot-side ESPs are equipped with a preheating system heating all internals to above 300°C before any gas is allowed to enter. A 300 mm mineral wool insulation ensures that the walls exposed to the gas during normal operating are maintained at high temperatures.

Due to the risk of occasional acid dew point peaks the internals as well as the casings of the cold-side ESPs are fabricated from AISI 316 stainless steel. Mild steel is used for the hot-side ESPs.

The Cooling Tower

The cooling tower is basically a standard Flakt design with downward gas flow. Water atomizing nozzles spraying in the gas flow direction are located just above the cylindrical part of the tower, figure 2. The inlet is designed to provide an even and stable gas distribution all the way through the tower. This ensures that the water is well mixed with the gas in order to obtain complete evaporation. The evaporation time depends mainly upon the water droplet sizes - average diameter as well as size distribution - and gas temperatures. Each tower is designed for a dry bottom - a complete evaporation - and for a gas flow of 20,000 Nm^3/h. Fine droplets are achieved by using air pressurized nozzles. Wear of the nozzles is minimized by using low operating pressures for the water and the air ($< 6 \times 10^5$ Pascal).

To meet the special process demands the cylindrical part of the tower consists of two concentric shells. Heated air in the annular space between the shells maintains the inner wall at fixed temperatures to avoid glass arsenic build-up. The upper part of the tower is kept at 300°C and the lower part at 150°C. The inner shell is fabricated of stainless steel, mainly AISI 316. The duct between the tower and the cold-side ESP is provided with equipment for steam conditioning to be used in case back corona effects are observed. The injection is automatically controlled from the electrical readings of the cold-side ESP.

Design Data

Design data for the dust collecting plant are given in table 3. Normal operation of the plant is based on equal gas flows in each circuit. However, the entire gas flow can be handled in one circuit with reduced collecting efficiency.

Table 3. DESIGN DATA FOR THE FULL-SCALE PLANT

	Stage 1 Hot-side ESPs	Stage 2 Cold-Side ESPs
Number of parallel ESPs	2	2
Number of fields/ESP	2	3
Total gas flow, Nm^3/h	40,000	46,000
Gas temperature range, ^oC	300 - 350	120 - 130
Nominal gas velocity in ESPs, two lines in operation, m/s	0.60	0.53
Inlet dust concentration, g/Nm^3	10	25
Guaranteed outlet dust concentration, g/Nm^3		
two circuits in operation	0.25	0.25
one circuit in operation	0.70	0.90

Operating Experiences

Selective collection of two separate dust fractions has been successful. However, two major operating problems have been experienced since the start-up of the plant. The first involved sticky dust at low temperatures - the second back corona phenomena. These problems were minimized in various ways.

Due to the sticky dust one circuit often had to be taken off-line for cleaning. During such periods the entire gas volume had to be cooled in just one cooling tower. This resulted in a wet bottom and dust build-up due to overloading. The problem was remedied with the installation of new air-pressurized nozzles. The bottom has since been dry.

Sticky dust and some corrosion in the cold-side ESP was caused by high concentrations of sulphuric acid generated in the roasting process. The excess of generated acid was reduced by increasing arsenic content of the concentrates and by lowering the temperature in the roasters. These steps eliminated problems of corrosion and sticky dust. At times, however, dust still builds up in the cold-side ESPs, requiring that a circuit be shut down after 3 - 4 months of operation. The build-up of As_2O_3 is removed manually using water. The collected As_2O_3 is of good quality and can be handled easily in the transporting system.

In figure 4, severe back corona conditions are contrasted with normal operating conditions in the cold-side ESPs by the current-voltage characteristics for different fields. The resistivity of the collected As_2O_3 dust has been measured at varying H_2SO_4 contents of the dust, figure 5. The dust samples contain about 95% As_2O_3 plus H_2SO_4 and 5% other compounds. Efficient ESP operation requires a certain range of resistivity levels. This means that only a limited variation of the H_2SO_4 contents - resulting in resistivities within the required range - is acceptable.

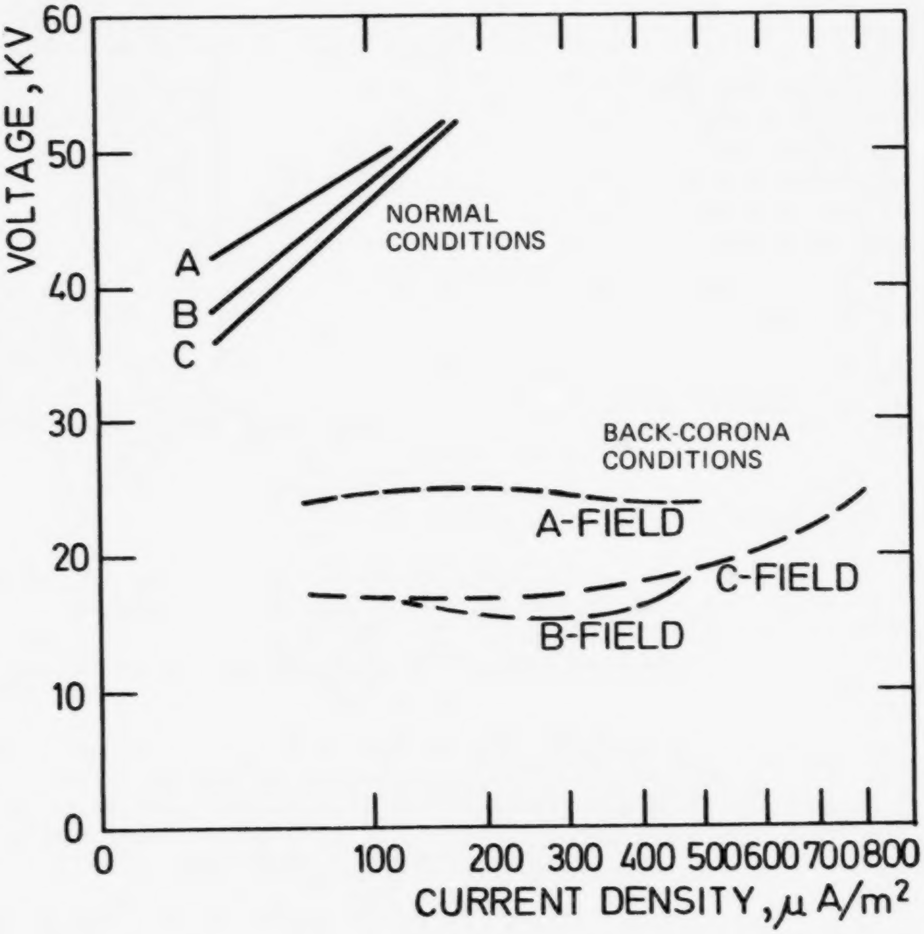

Fig. 4 Current-voltage curves for the cold-side ESPs

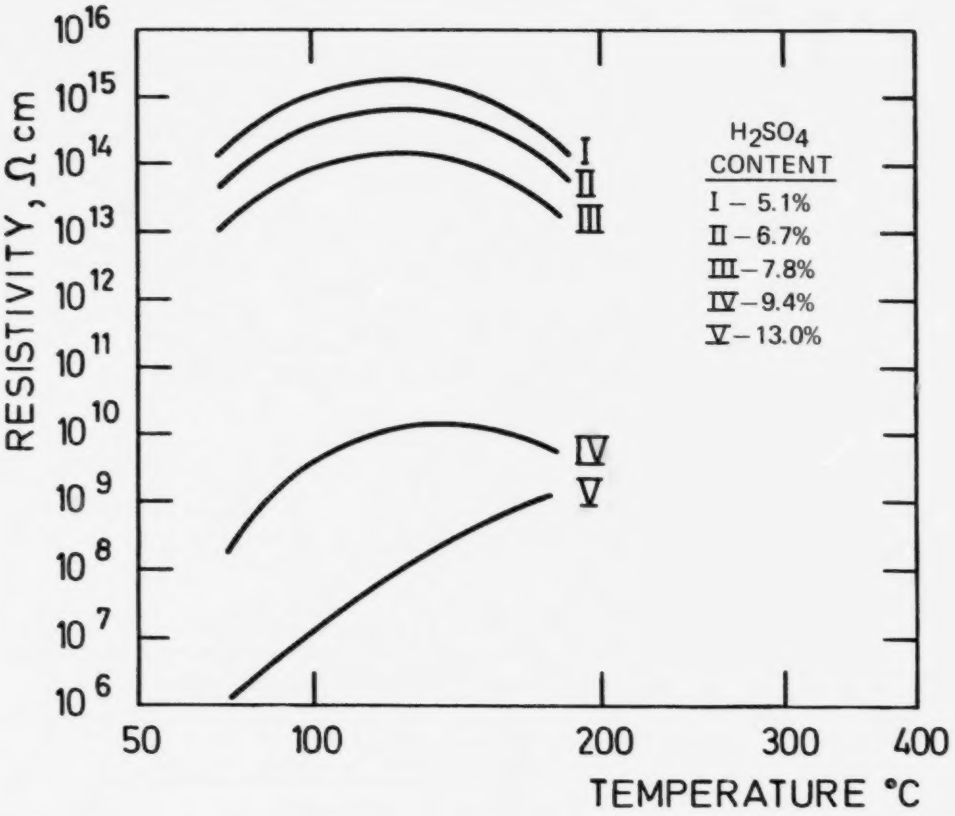

Fig. 5 Resistivity of As_2O_3 as a function of temperature and H_2SO_4-content of the dust

It is preferable to operate with a moderate H_2SO_4 content in the dust to avoid sticky dust in the event of increased SO_3 generation resulting from disturbances in the roasting process. Thus back corona conditions will still occur occasionally.

Analyses of the collected dusts are shown in table 4. These values, as well as those given in tables 5 - 8, are mean values for one year and are based on operation with the entire gas volume through one circuit. Dust from the cold-side ESP often contains approx. 65% As, corresponding to 85% As_2O_3.

The relative distribution of different elements in the two stages is shown in table 5. The total recovery of copper, zinc, lead and arsenic has been excellent, tables 6 - 8. The recovery of arsenic trioxide in the ESPs under normal operating conditions approximates 97% (for comparison, see table 7). The corresponding concentration of As_2O_3 (solids and gaseous) after the ESPs is less than 250 mg/Nm^3.

Table 4. ANALYSIS OF COLLECTED DUST IN HOT-SIDE AND COLD-SIDE ESPS

	S %	Fe %	Cu %	Pb %	Zn %	As %	Hg g/ton
Hot-side ESP	11.2	19.6	15.7	7.0	5.6	5.6	75
Cold-side ESP	6.3	0.21	0.16	0.18	0.09	55.1	3200

Table 5. RELATIVE DISTRIBUTION OF ELEMENTS BETWEEN HOT-SIDE AND COLD-SIDE ESPS

	S	Fe	Cu	Pb	Zn	As	Hg
Hot-side ESP,%	75.1	99.4	99.4	98.5	99.1	14.7	3.8
Cold-side ESP, %	24.9	0.6	0.6	1.5	0.9	85.3	96.2

Table 6. RECOVERY OF ELEMENTS IN THE INSTALLATION INCLUDING THE WET STEP

	Cu	Pb	Zn	As	Hg
Total recovery, %	99.96	99.91	99.73	97.81	84.02

Table 7. RECOVERY OF ELEMENTS IN THE HOT-SIDE AND COLD-SIDE ESPS

	Cu	Pb	Zn	As	Hg
Recovery, %	99.91	99.63	98.71	88.69	69.04

Table 8. CONCENTRATIONS IN THE GAS AFTER THE COLD-SIDE ESP

	Hg as dust	Hg as vapour	As as dust and vapour
mg/Nm^3	4.5	4	470

Installation cost of the full-scale plant approximated 20 million Swedish Crowns. Operating cost is about 4 million Swedish Crowns a year, including labour, electricity, steam, water treatment and maintenance. In arriving at the above figures, no consideration was given to the value of collected metals. In view of the recovery value of the metals, the incremental cost of the hot-side ESPs is justified.

Future Operation

Plans call for the multi-hearth furnaces to be replaced by a fluosolid furnace within two years. The dust collection plant will remain unchanged. SO_3 generation from the new furnace should be easier to control thereby assuring stable conditions in the cold-side ESP.

References

1. Matts, S. Staubwiderstand und Wahl der Sprühelektroden. VDIBerichte Nr.294, 1978.

2. Maartmann, S. The Effect of Gas Temperature and Dew Point on Dust Resisitivity and thus the Collecting Efficiency of Electrostatic Precipitators. Academic Press, New York, 1971. Proceedings of the Second International Clean Air Congress.

THE WSL APPROACH TO METALLURGICAL FUME EMISSIONS

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ABSTRACT

Research is in progress at Warren Spring Laboratory (WSL) to reduce the quantity of metal lost as fume and to minimise the energy required for its collection in pyrometallurgical operations. The paper is in three parts:

(1) a brief discussion of the mechanisms of fume formation and growth;

(2) a description of the research into fume suppression using flux covers, particularly for brass melting;

(3) a review of the available agglomeration techniques and WSL's work in this field.

INTRODUCTION

Metallurgical fume emissions make a significant contribution to pyrometallurgical processing costs for two reasons. Firstly, valuable metal is lost and often has to be replaced by high cost virgin metal which, in the UK, is almost certainly imported either as concentrate for processing or as metal. Metal losses reduce production rates and hence profitability. Secondly, fume emissions have to be controlled to protect the environment and/or to recover valuable metals such as tin, silver and gold. Consequently, costly gas cleaning plants have to be installed and these are often expensive to run both in terms of energy requirements and maintenance. For these reasons a research programme was started at WSL to investigate methods of reducing the cost of fume emissions in pyrometallurgical processes, particularly by suppression and agglomeration.

MECHANISMS OF FUME FORMATION

An understanding of the mechanisms of fume formation is vital to the WSL programme, particularly with respect to particle growth. A comprehensive report covering formation mechanisms has been produced by Ollerenshaw[1] and other publications by Fuchs[2], Ramabhadran et al[3], Hidy and Brock[4] and Buckle[5] are cited here. The fuming process begins with vaporisation of the more volatile metals such as zinc, lead and cadmium in metal melting and refining operations. The various stages of this process are outlined briefly in Figure 1.

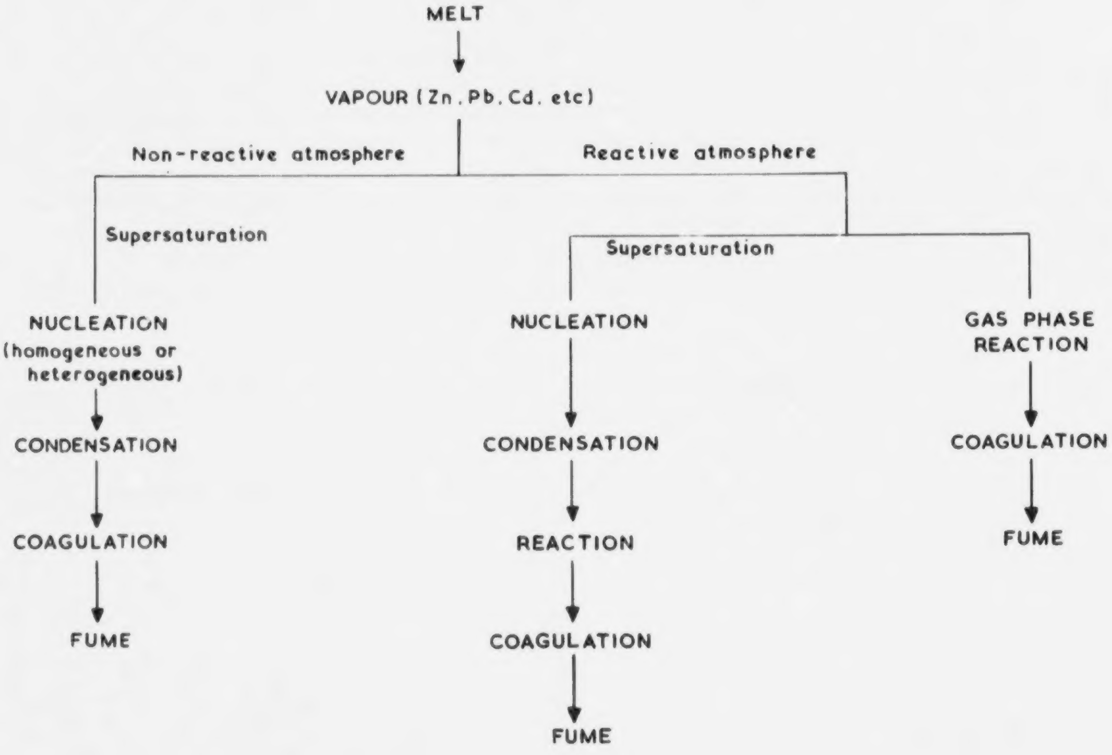
MELT
VAPOUR (Zn. Pb. Cd. etc)
Non-reactive atmosphere
Reactive atmosphere
Supersaturation
Supersaturation
NUCLEATION (homogeneous or heterogeneous)
CONDENSATION
COAGULATION
FUME
NUCLEATION
CONDENSATION
REACTION
COAGULATION
FUME
GAS PHASE REACTION
COAGULATION
FUME

Fig. 1 Mechanisms of fume formation

In an inert atmosphere, nucleation occurs as supersaturation of the vapour phase is reached. This can be either homogeneous or heterogeneous, the latter occurring when foreign particles such as dust etc are present. The nuclei so formed act as vapour 'sinks' and grow by condensation of vapour onto their surfaces. Further growth by coagulation results from collisions between particles/droplets. With a gradual decrease in temperature, substantial growth can be achieved chiefly by condensation as, for example, in metal powder production. Similar mechanisms apply to volatile metal oxides, chlorides etc such as those of lead and antimony.

In reactive atmospheres two formation mechanisms are possible depending on whether reaction occurs before or after nucleation. This in turn depends mainly on the temperature gradient and gas composition near the melt surface. When the vapour nucleates as metal, growth by condensation can proceed and hence some control of particle size is possible. However, if nucleation as oxide is the predominant mechanism growth by condensation usually cannot occur because many oxides, particularly the refractory ones such as zinc oxide, vaporise only at very high temperatures (>2000°C) by disproportionation. Consequently the only significant growth mechanism is coagulation as a result of Brownian diffusion and this effect becomes insignificant as the partcle size reaches about 1 μm. Thus there is no control of particle size for this mechanism.

In practice most metallurgical fumes are produced in reactive atmospheres, the most commonly arising being zinc oxide. Almost every sample of zinc oxide fume characterised in the laboratory has been extremely fine with a mean particle size below 1 μm, which suggests formation by nucleation as oxide. However, one sample taken from a rotary kiln operation to remove zinc and lead from steelmaking fume was found to be much coarser as shown in Figure 2. As reducing conditions existed whilst the metal was in the vapour phase, it is possible that nucleation occurred before oxidation and hence growth by condensation was responsible for the observed increased particle size.

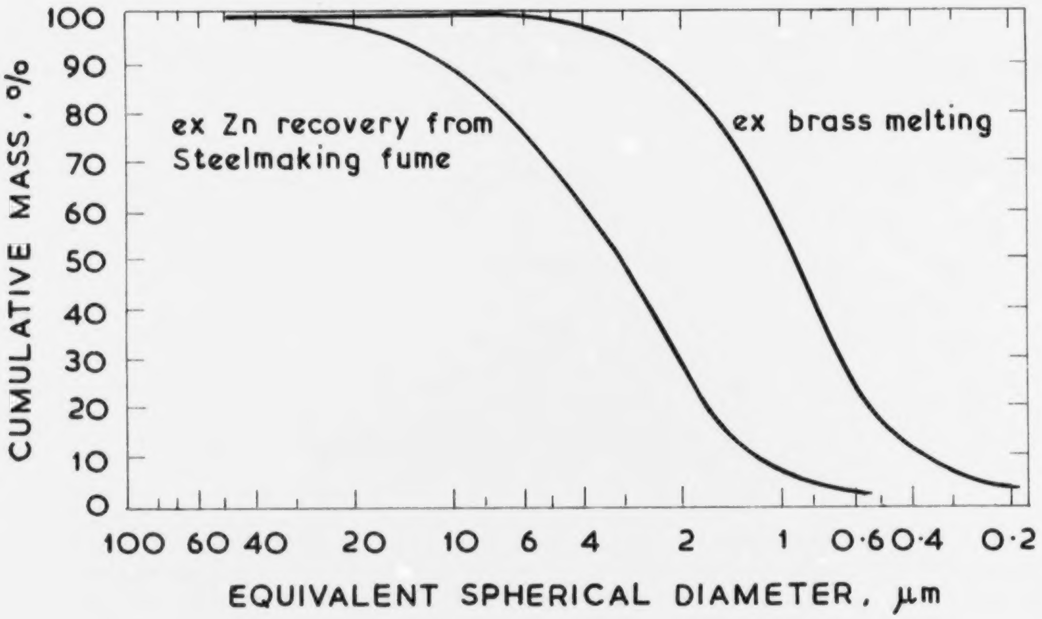

Fig.2 Particle size distributions for ZnO fumes

In most practical situations control of temperature gradients and atmosphere composition is not possible and particle formation occurs in a narrow boundary layer close to the melt surface. Hence there is little likelihood of particle size control at the source of generation. For this reason attempts have been made to find alternative solutions to the fume problem and work on suppression and agglomeration have featured prominently in WSL's programme.

FUME SUPPRESSION

Fume emission rates are dependent on the vaporisation rate from the melt surface and can be reduced by using a flux or slag cover. Secondary brass production using an induction furnace was selected for laboratory studies in this field and comprehensive reports have been published elsewhere [6,7]. Flux covers are often used in this industry to suppress zinc oxide fume emissions.

Procedure

A 50 kW, medium frequency, coreless induction furnace was used to melt approximately 27 kg of 60/40 brass scrap consisting of swarf, stampings, strip and ingot. The fume generated was collected in a small bag filter unit as shown in Figure 3.

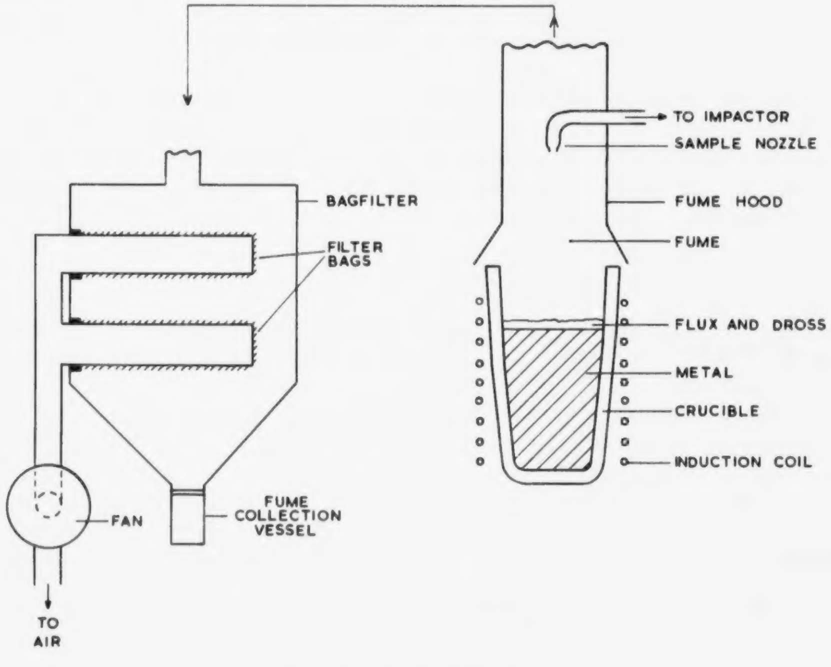
TO IMPACTOR
SAMPLE NOZZLE
BAGFILTER
FUME HOOD
FILTER BAGS
FUME
FLUX AND DROSS
METAL
CRUCIBLE
INDUCTION COIL
FAN
FUME COLLECTION VESSEL
TO AIR

Fig. 3 Schematic of Apparatus

The melting procedure simulated typical plant practice. Scrap was charged to the furnace in the order swarf, zinc, stampings (or strip), ingot and flux, and melted into a molten heel of metal. A typical charge is given in Table 1.

Table 1. TYPICAL CHARGE COMPOSITION

	kg	%
Heel	11.17	40.8
Swarf	4.54	16.6
Cuttings	5.39	19.7
Ingots	5.58	20.4
Zinc	0.49	1.8
Flux	0.20	0.7
	27.37	100.0

The metal was heated to 1080°C and then the dross was removed before pouring into three ingot moulds. A selection of commercial and non-proprietary fluxes was used, the latter consisting of borax, borosilicates, pyrex and cullet glass, and sodium carbonate.

All products were analysed chemically for copper, zinc and lead. Drosses were rod-milled and screened at 850 μm (18 mesh), the oversize being recycled as metal whilst the undersize was chemically analysed. Fume particle size distributions were determined using a cascade impactor and also a Sedigraph 5000 D analyser.

Results

The results have been described in detail elsewhere[6,7] and are summarised briefly in Table 2. The following observations can be made:-

(1) Fume emissions were reduced by up to 40% by some fluxes but their size distributions were very similar.

(2) Dross losses varied considerably from 1.8 to 3.5% of the charge weight, but their metal contents were similar except when liquid fluxes (borax based) were used, when metal entrainment was greatly reduced.

(3) The flux had no effect on the chemical composition of the metal.

Table 2. THE EFFECT OF FLUXES IN A BRASS MELTING OPERATION

Flux	Fume Loss, % Charge	Dross Loss, % Charge	Cu Content of Dross, %	Metal Entrained, % Charge
None	0.17	5.05	23.0	1.93
A	0.22	2.01	22.2	0.75
B	0.10	2.66	18.5	0.82
C	0.10	2.94	18.9	0.92
D	0.33	2.33	14.8	0.57
E	0.12	2.33	12.9	0.50
F	0.14	3.01	22.1	1.10
G	0.12	2.64	25.8	1.13
H	0.18	2.65	17.2	0.77
I	0.27	2.16	14.7	0.53
J	0.14	2.73	23.1	1.05
Borax	0.11	1.86	4.5	0.13
Borosilicate	0.11	2.10	4.7	0.16
Pyrex	0.22	3.43	27.7	1.58
Cullet	0.20	3.71	28.9	1.78
Sod. Carb.	0.06	2.86	14.2	0.70

Flux Classification

The fluxes used in this work are classified in Table 3. They can be categorised into three types as follows:

(1) Powdery, reducing fluxes containing mostly charcoal - these fluxes formed a porous layer through which vapour escaped readily and high fume emissions were recorded. Metal entrainment was also high.

(2) Hard crust - this group consisted of the remaining commercial fluxes. They melted at low temperatures but re-solidified before the end of the melt cycle due either to loss of water of crystallisation or to uptake of zinc oxide as reported by Deev et al[8]. The crust so formed could not be penetrated by zinc vapour from the melt except at weak points where the flux met the crucible wall. Here vapour burnt and gradually built up tubes sometimes growing as long as 5-10 cm, as shown in Figure 4. Most of the metal loss was swarf and other unmelted scrap entrained in the dross layer.

(3) Liquid layers - fluxes which melted and remained liquid until the end of the melt cycle were most effective. The low viscosity compounds covered the entire melt surface forming a most effective barrier to vapour escape. As the pressure built up, bubbles formed and broke through the layer oxidising rapidly as they burst. This phenomena has also been

reported by Goloveshko[9]. Metal entrainment was greatly reduced because heavy metal particles sank through the liquid layer. Viscous fluxes were less effective because they did not cover the melt but formed a sticky mass, which entrained a high proportion of metal. Consequently fume loadings and metal losses were very high for these compounds.

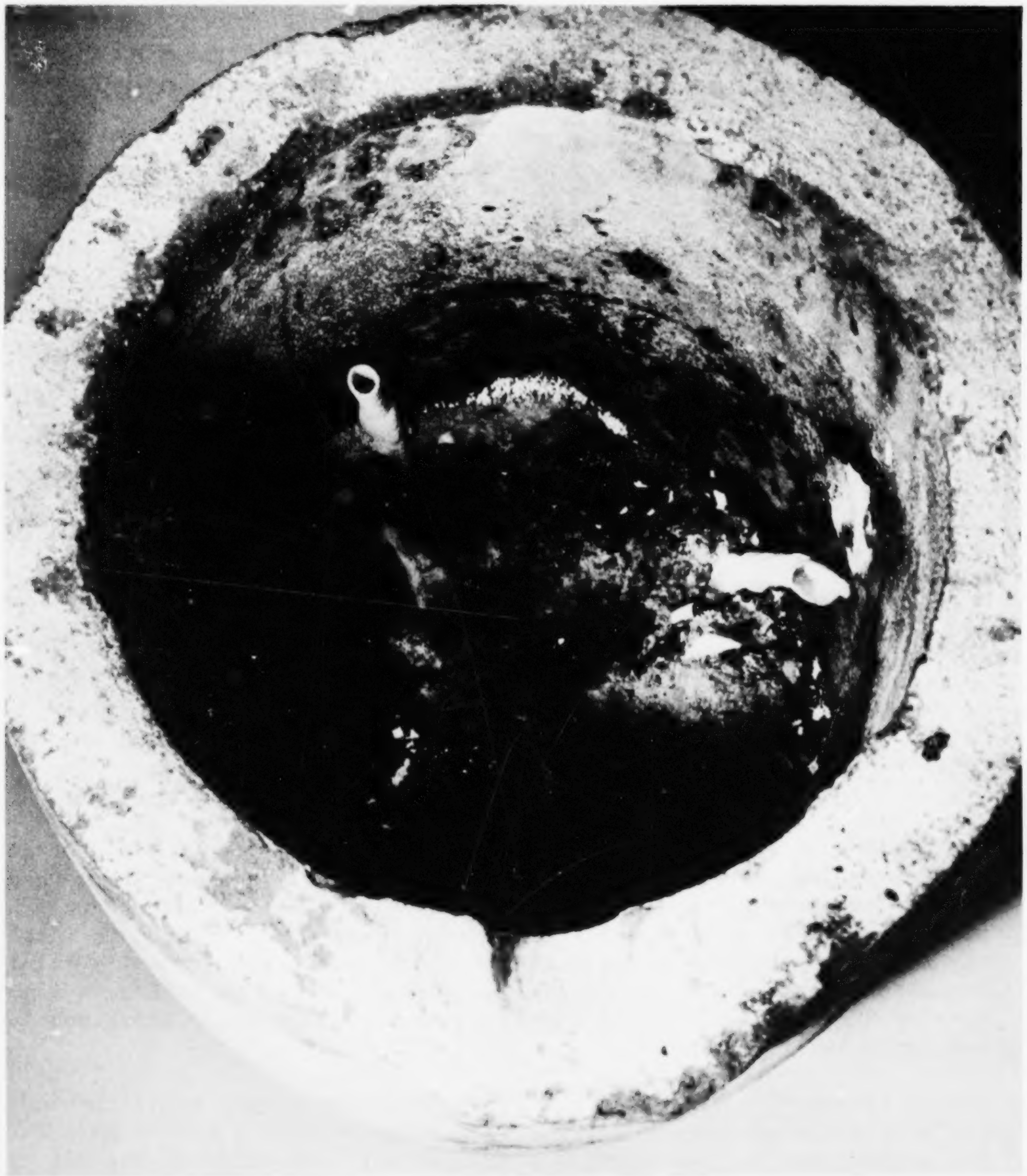

Fig. 4 Development of ZnO spirals from melt surface

Table 3. FLUX CLASSIFICATION

Category	Fume Emission	Dross Loss	Fluxes in Category
1. Powdery, sometimes reducing	H	H	A,D
2. Flux melts and resolidifies giving hard crust.			
(a) Non-volatile flux	M	M	B,C,E,F,G,H,J,K
(b) Volatile flux	H	L	I
3. Liquid layer			
(a) Low viscosity	L	L	Borax, BS2
(b) High viscosity	H	H	Pyrex, Cullet

Approximate ranges (in % of charge weight)

	L	M	H
Fume	<0.1	0.1-0.2	>0.2
Dross	<2.2	2.2-3.0	>3.0

The main problem with molten fluxes is removal prior to metal pouring. Conventionally the drosses are solid and are raked off but this is impossible with liquid drosses. They are not easily poured off either. They can be thickened or 'killed' with sand but this creates large quantities of dross for subsequent handling, treatment and recycle. The ideal flux should be liquid through the melt cycle, solidifying only at the end.

Research at WSL has shown that mixtures of sodium carbonate and borax make suitable fluxes. As Figure 5 shows, the optimum composition is roughly 50% each of borax and sodium carbonate and this is the minimum level of the latter if a solid dross is required.

AGGLOMERATION

The use of fluxes is restricted generally to remelting processes. Clearly they are undesirable in refining processes where impurities such as zinc, cadmium etc are deliberately fumed off and so techniques to facilitate fume collection have been studied, notably by agglomeration. Extensive reviews on this subject have been presented by Dawson[10,11], Hegarty and Shannon[12] and Ananth and Shannon[13-15].

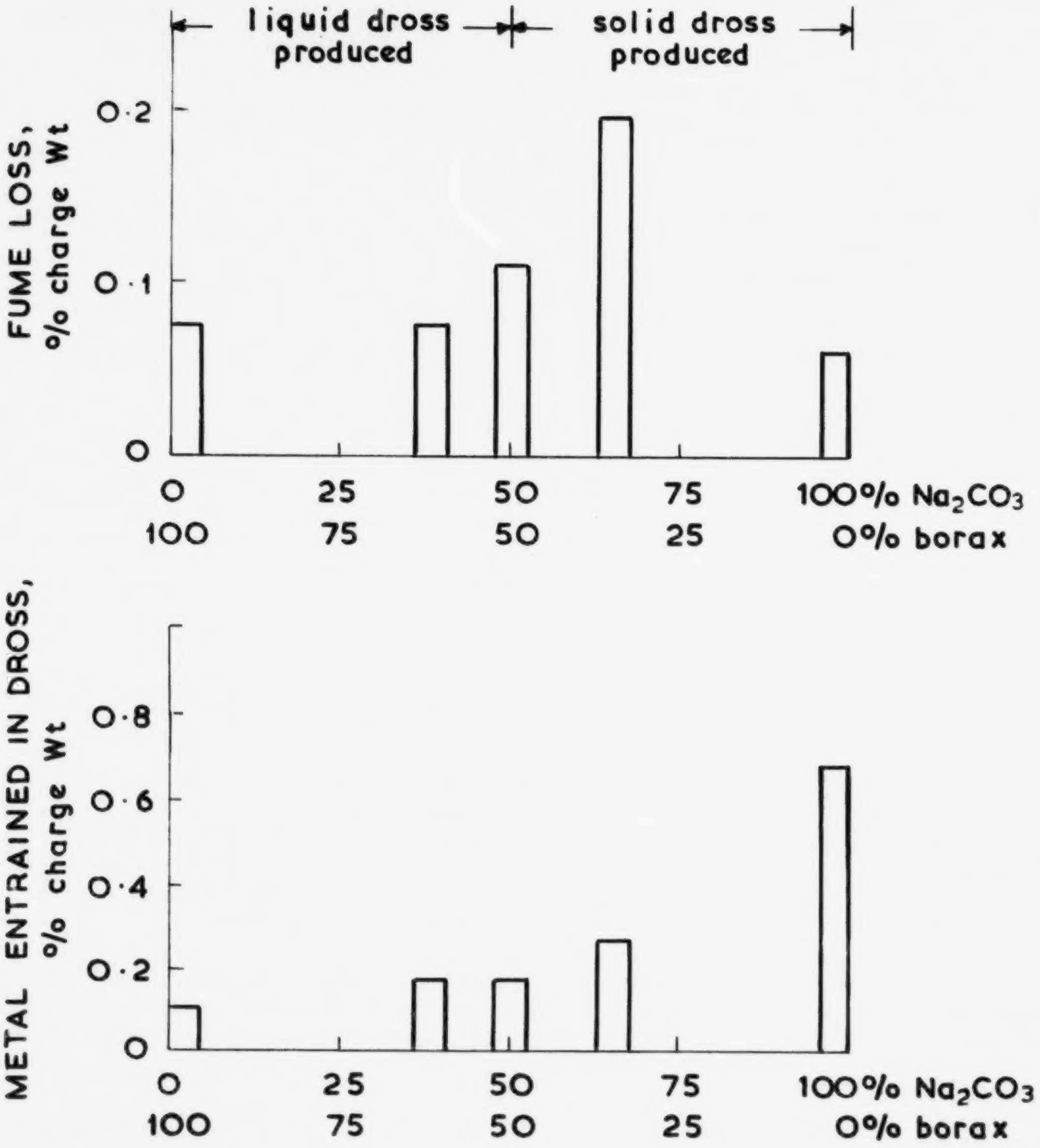

Fig. 5 Fume and metal losses from a brass melting operation using borax / Na_2CO_3 fluxes

Energy Requirements in Gas Cleaning

Gas cleaning is a major contributor to the capital and operating costs of many pyrometallurgical processes. In particular the energy requirements can be very high and can be related to the fume particle size as shown by Teller[16]. As the power requirement rises exponentially with decreasing particle size, submicron fumes present the greatest problems in terms of collection efficiency and cost. Only high efficiency collection plants are effective for control of such fumes and high capital and operating costs are incurred. However, by agglomerating 0.1 μm particles to 1 μm particles, for example, power requirements could be reduced by a factor of 6 or 7 according to Teller[16].

Agglomeration can also be used to uprate existing gas cleaning plants. As environmental legislation has become more stringent, the demands on gas cleaning plant efficiencies have increased dramatically. For wet scrubbers, in particular, increased efficiencies can be achieved by utilising more powerful fans or by changing venturi geometries etc but this increases the power requirements. Moreover, electricity consumption[17] increases exponentially for efficiencies greater than 95%. Consequently a critical economic situation can be reached when it may be cheaper in the long run to build new plant rather than incur the increased energy costs. Agglomeration offers a third choice.

Theoretical Considerations

Agglomeration of small particles suspended in a gas stream occurs when they collide and coalesce. The basic coagulation equation is given by Hidy and Brock[4]

$$\frac{dn_k}{dt} = G_k = \frac{1}{2}\left[\sum_{\substack{j=1\\ i=k-j}}^{i=k-1} b_{ij}\, n_i\, n_j\right] - n_k\left[\sum_{i=1}^{\infty} b_{ik}\, n_i\right] \qquad ..... (1)$$

where the first term on the right hand side represents the gain to species k by collisions between i and j particles, while the second term denotes the loss by collisions of k species and all other particles.

Using the definition of the continuous spectrum function n(v,t), eqn (1) can be rewritten as an integro-differential equation as follows:

$$\frac{\partial n}{\partial t} = \frac{1}{2}\int_0^v b(m,n)\, n(m,t)\, n(v-m,t)dm - n(v,t)\int_0^\infty b(m,v)\, n(m,t)dm \qquad ..... (2)$$

The collision frequency term b(m,n) is of major significance in agglomeration studies. In some cases coagulation is a spontaneous process due to Brownian diffusion. It can be shown that provided particle diameter d is much greater than λ, the mean free path of surrounding gas atoms,

$$b(m,n) = 4\pi (D_1 + D_2)(r_1 + r_2) \qquad \text{..... (3)}$$

where r_1 is the radius of particle of volume m, and r_2 that of particle of volume n. The diffusion coefficient of particle i in the gas is given by (Kennard[18]):

$$D_i = \frac{kT}{6\pi\eta r_i} \qquad \text{..... (4)}$$

At N.T.P., the mean free path of a gas atom is about 0.1 μm which could be greater than some particle diameters. In this case Williams[19] gives the following equation for collision frequency:

$$b(m,n) = \left(\frac{8\pi kT}{\rho_p}\right)^{1/2} \left(\frac{3}{4\pi}\right)^{2/3} \left(\frac{m+n}{mn}\right)^{1/2} (m^{1/3} + n^{1/3})^2 \qquad \text{..... (5)}$$

Agglomeration due to Brownian motion is inversely dependant on particle size and consequently is less effective for larger particles. In practice it is insignificant for particles larger than 1 μm and only of minor importance for those larger than 0.5 μm.

Agglomeration rates can be increased by increasing the collision frequency b(m,n). This can be done by applying external fields such as sonic or electric/magnetic, by creating turbulence, or by increasing the number of collision sites as in steam condensation and a fluidized bed.

Sonic Agglomeration

Collisions can occur between particles of different size or density moving at different velocities as a result of vibrations induced by sound waves. The smaller particles take part in the sound wave vibrations with larger amplitude due to their small mass and they collide with larger particles which hardly vibrate at all. The ratio between particle and gas amplitude is given by Williams[19] as:

$$\frac{A_p}{A_g} = \frac{K}{\sqrt{K^2 + \omega^2}} \qquad \text{..... (6)}$$

where $K = \dfrac{9\eta}{2r^2\rho}$

For water droplets in air at NTP the ratios were 0.93 and 0.39 for 1 μm and 2 μm particles respectively at the same frequency. This shows that quite small differences in mass led to very different amplitudes and strong phase differences. Hence acoustic excitation of a fine particle suspension could increase the agglomeration rate.

An expression for coagulation rate was derived by Mednikov[20]:

$$\frac{dn}{dt} = - F n_s \qquad \text{..... (7)}$$

If the particle count at time t = 0 is n_o, then the solution to eqn (7) is

$$n = n_o e^{-Ft} \qquad \text{..... (8)}$$

Hegarty et al[12] derived the following relationship for F:

$$F = 2 \pi \beta n_L \varepsilon r_L^2 a A \qquad \text{..... (9)}$$

A can be related to sound intensity J according to:

$$A = \frac{1}{\omega} \sqrt{\frac{2 J}{\rho_g C}} \qquad \text{..... (10)}$$

Hence agglomeration rate is dependent on frequency, particle number density and size distribution, sound intensity and exposure time.

To achieve significant growth, high intensities have to be used. Hegarty and Shannon[12] related residence time and intensity to a growth ratio r_f/r_o:

$$t J^{\frac{1}{2}} \simeq 3 \ln \frac{r_f}{r_o} \qquad \text{..... (11)}$$

For 5-20 fold growth, intensities of 150-165 dB would be needed with corresponding residence times of 6-200 seconds. Hegarty and Shannon also calculate that the energy requirement would be at least 53.6 kW per 100 m^3 min^{-1} for a 4 m chamber operating at 160 dB. Mednikov[20] estimated energy requirements between 10.7 and 26.8 kW per 100 m^3 min^{-1} air flow. These energy levels are very high, greater than many gas cleaning plants and severely restrict the application of sonic agglomeration.

Agglomeration in Laminar and Turbulent Flow Fields

Agglomeration can be promoted by flow fields due to the relative velocity between particles. Smoluchowski[21] has shown that in laminar flow, the collision rate is given by:

$$b(m,n) = \frac{4}{3} G (r_1 + r_2)^3 \qquad \text{..... (12)}$$

where G is the constant shear rate = $\frac{dU}{dx}$.

Levich[22] derived the following equation for agglomeration rate, Ψ, in turbulent flow:

$$\Psi = 32\,\pi\, r^3\, F\, n_o = 25 \left(\frac{e}{\eta}\right) r^3\, n_o \qquad \text{..... (13)}$$

A similar equation was derived by Beal[23] who included Brownian motion in Levich's theory:

$$\Psi = Q\, \lambda_o^{2/3} \left(\frac{e}{\eta}\right)^{1/2} r^{1/3}\, n_o \qquad \text{..... (14)}$$

where λ_o is the microscale of turbulence = $(\eta^3/e)^{1/4}$.

The agglomeration rate is strongly dependent on particle size and decreases rapidly for submicron particles. It can be increased by increasing the gas velocity but this raises the energy requirements. Ananth and Shannon calculated that 107.2 kW per 100 m^3 min^{-1} air flow might be necessary to agglomerate 0.5 μm particles to 2-3 μm. Even if this energy level could be tolerated, the time required for growth would be about a minute which is unacceptable.

Agglomeration due to Applied Electric/Magnetic Fields

Externally applied electric/magnetic fields induce dipole moments on particles resulting in interactions and possible agglomeration. Fuchs[2] showed that the mean value for the ratio of agglomeration constants for charged and uncharged particles, $\bar{z}$ is given by:

$$\bar{z} = \frac{1}{12}\,\frac{\bar{E}^2\, r^3}{kT} \qquad \text{..... (15)}$$

The agglomeration rate is thus strongly dependent on field intensity and particle size. Magnetic fields are by far the stronger of the two and can be applied to much smaller particles. However as the rate is proportional to r^3, this technique is of little value in agglomeration of submicron particles. Ananth and Shannon[15] also point out that magnetic fields can only be applied to ferromagnetic particles which limits their use considerably.

Steam Condensation

Collision frequency can be increased by injecting foreign particles or droplets into the fume laden gas. This can be achieved by injecting steam into the particle suspension and allowing it to condense. Schauer[24] collected 99% of 0.3 μm diameter dioctyl phthalate particles in a wet scrubber after injecting steam where previously collection had been very poor. Prakash and Murray[25] claimed a tenfold increase in particle size by using up to 10 kg steam per 100 m^3 air. Steam production is expensive and such injection levels would be costly if waste heat is not available.

Steam injection is particularly attractive as an agglomeration technique prior to a wet scrubber. Calvert and Jhaveri [26] have reported promising results for a flux force/condensation (FFC) scrubber. They show that the scrubber collection efficiency increases with increasing vapour condensation. The FFC method becomes attractive economically when high collection efficiencies of submicron particles are sought. These are the most difficult and the most energy consuming operating conditions for the conventional wet scrubber. Calvert and Jhaveri[26] also suggest that this technique could be suitable for uprating existing scrubbers, but in all cases, the cost of the steam is a major consideration.

Fluidized Bed Agglomeration

The use of fluidized beds for fume agglomeration has not previously been described although Pilney and Erickson[27] reported that flyash particles in the effluent air stream after fluidized bed filtration were in fact agglomerates. Conventionally fluidized beds have been used to remove dust and aerosol particles for some time, the earliest reported work being that of Meissner and Mickley[28] in 1949. As a collection device, it has found applications in primary aluminium production as the Alcoa 398 process[29], and in asphalt recycling[30].

The most serious problem in fluidized bed filtration is retention of the fume once it has impacted. Usually to avoid excessive re-entrainment the bed is changed continually and the used collection media can be either dumped, recycled (as in Alcoa 398) or cleaned for recycle. Particles within a fluidized bed collide or rub against their neighbours (or the walls) and consequently it is probable that surface layers of collected fume will break off. These new species will be agglomerates having a significantly larger particle size than the inlet fume. Hence if the bed is not changed and layers of fume are allowed to build up on the collector particles, it could operate as an agglomerator and this is the basis of WSL'S fluidized bed agglomerator.

According to Jackson [31], the principal collection mechanisms are Brownian diffusion for particles smaller than 0.5 μm, inertial impaction for particles larger than 2 μm, and interception in the intermediate region. Knettig and Beeckmans [32] and Jackson[31] concluded that most collection (or collisions) occurred near the distributor. Doganoglu et al[33] explained the effect of gas velocity on collection efficiency. They used a multi-orifice distributor with low free area (1.5-6.7%) and related collection efficiency with an impaction parameter or Stokes number, St, given by:

$$St = \frac{C \rho_a d_a^2 U_{or}}{9 \eta d_p} \qquad \text{..... (16)}$$

where U_{or} is the gas velocity through a distributor orifice.

Most collection occurs in the grid jet region. With small grid jets, orifice velocities are high and fume particles impact on the bed particles there. High collection efficiencies result as found by Meissner and Mickley[28], Knettig and Beeckmans[32], Doganoglu et al[33], and Chaturvedi and Reed[34]. On the other hand when an open perforated plate is used as a distributor, the orifice velocity U_{or} is too low to gain advantage of inertial collection in the grid jets.

Using small diameter grid jets, small bubbles form and they coalesce rapidly giving rapid transfer between the bubble and particulate phases. In contrast using the more open distributors, larger bubbles are formed and interphase transfer is slower. Bypassing is common and consequently the collection efficiency decreases as gas velocity increases.

It is possible for impaction to dominate collection in the grid jet region whilst diffusion dominates in the bed itself. If this is the case then collection efficiency should increase with deeper beds. This has been confirmed experimentally by Doganoglu et al[33] and by Chaturvedi and Reed[34].

As has been pointed out, retention of the fume on collector particles may be the critical factor, for it is by no means certain that two such particles will stick together after collision. If agglomeration is the aim, however, retention is less important because re-entrainment is part of the process. The physics of retention have not been studied closely. Claes et al[35] and Dumont et al[36] have examined the retention of soot particles on 150 and 220 μm alumina particles. They found that the bed could accommodate about 0.05% of its mass of fume before efficiency declined.

Retention can be improved considerably by incorporating a liquid binder to render the collector 'sticky' to dust or fume particles. Doganoglu et al[33] coated their bed particles with a non-volatile liquid (dioctyl phthalate) and this greatly improved retention. Pilney and Erickson[27] showed that fly ash particles could be filtered in a fluidized bed when a small amount of water was introduced in the bed, but there are practical difficulties in the use of such collection aids.

Anderson and Silverman[37], and Jugel et al[38] examined the effect of electrostatics in fluidized beds and retention therein. Bed particles become charged due to friction with their neighbours and the walls of the bed (triboelectrification). Anderson and Silverman[37], and Jugel et al[38] have shown that this improves retention but few details are given. Ciborowski and Zakowski[39] showed that the effect decreased as air humidity increased. Zahedi and Melcher[40,41] have reported high collection efficiencies using their electrofluidized bed.

Theoretically, fluidized beds incorporating low free-area distributor plates are capable of the highest collection because the high orifice velocities promote inertial impaction - the dominant collection mechanism. A free area of about 3% appears reasonable from WSL's work. However, as the collision frequency is particle size dependent, higher orifice velocities are needed for smaller particles and the pressure drop across the distributor plate is thus higher. For submicron particles a pressure drop of about 5 ins w.g. is indicated, which is roughly the same as that of a bag filter or electrostatic precipitator but much less than that of a high efficiency venturi scrubber.

It is difficult to convert a pressure drop to energy terms because factors such as fan efficiency, rotor speed, type of impeller etc must be considered. However, the increased fan speed is related to absorbed horse power according to:

$$\frac{HP_1}{HP_2} \propto \left(\frac{N_1}{N_2}\right)^3 \qquad \text{..... (17)}$$

for a particular fan assuming constant air density. The proportionality is based on a constant fan efficiency but this can also vary in practice and might be improved using a different impeller. Hence, if used to uprate existing plants, the fluidized bed agglomerator would be most attractive for those incurring high operating costs, e.g. high efficiency venturi scrubbers.

A different situation exists when installing new plant. The capital costs of bag filters and electrostatic precipitators are high, much higher than for a fluidized bed, for example. Hence a combination of agglomerator and low capital cost plant, such as low efficiency venturi, cyclone etc, might also be considered on economic grounds, assuming that comparable efficiency can be attained.

WSL has commenced work on fluidized bed agglomeration of zinc oxide fume. So far the bed has been operated in the collection mode only using 250 μm and 500 μm ballotini particles and roughly 80% collection efficiencies have been obtained. The next step will be continuous operation, monitoring fume particle size and loading before and after the bed to assess retention and agglomeration potential.

CONCLUSION

WSL's fume programme has been in existence for three years during which time fume formation mechanisms have been reviewed extensively and two key areas of research have emerged - suppression and agglomeration. Fluxes for use in the secondary brass industry have been classified into three groups and a suitable flux has been prepared to minimise metal losses in fume and dross.

Fume agglomeration techniques have been reviewed and their potential for reducing gas cleaning costs has been assessed. Agglomeration rate is dependent on collision frequency and can be increased by increasing the number concentration of particles as in steam condensation, and using a fluidized bed. Techniques using applied fields have been discounted because of the high energy requirements and/or long reaction times.

NOTATION

A	amplitude of displacement
a	entrainment factor
b, b_{ij}	collision parameters
$b(m,n)$	collision frequency between drops or particles of volumes m and n
C	Cunningham slip factor
D	diffusion coefficient
d	diameter
E	electric field intensity
e	energy dissipated per gram second
F	coagulation constant
f	frequency
J	sound intensity
k	Boltzmann's constant
n_i, n_j, n_k	number density of particles of species i,j,k
$n(m,t)$	particle volume distribution at time t
Q	constant
r	radius
T	absolute temperature
t	time
U	gas velocity
V	volumetric flow rate

Greek Letters

ε	radius of assumed cylindrical coagulation zone
λ	mean free path of gas atoms
η	gas viscosity
ω	angular frequency of sound wave = $2\pi f$
ρ	density

Subscripts

a	aerosol or fume particle
f	final
g	gasesous phase
i,j,k	index for particles of species, i,j, or k
L	large particle
o	initial
p	collector particle
s	small particle

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ADVANCED TECHNIQUES FOR FINE PARTICULATE CONTROL

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ABSTRACT

The Particulate Technology Branch of the U. S. Environmental Protection Agency's Industrial Environmental Research Laboratory, located at Research Triangle Park, North Carolina, has for the past 5 years had the major responsibility for carrying out a research and development effort aimed at evaluating existing novel devices for fine particulate control, and discovering and bringing to a commercial feasibility stage devices based on new collection principles or concepts or new combinations of existing concepts. This paper presents test results of the concepts and devices which have been developed and/or tested by EPA. Emphasis is placed on those concepts or devices which may find application in the metallurgical industry.

INTRODUCTION

Particulate emissions are usually collected from an industrial smoke stack by one or more types of three "conventional" devices--fabric filters, wet scrubbers, and electrostatic precipitators. The design of each of these three devices has until recently remained relatively unchanged since the turn of the century. In recent years because of the need to collect smaller and smaller particles, meaning higher and higher collection efficiences, many improvements have been developed for the "conventional" devices, and a number of novel devices and/or concepts have been proposed, developed, and/or marketed.

As the requirement to collect finer and finer particulate has been imposed, the cost of conventional particulate control has risen. Since many important collection mechanisms become much less effective on fine particles (particles less than 1-3 micrometers in diameter), conventional devices (with the exception of fabric filters) have become larger or have required more energy to operate and thus have become more expensive. In order to minimize the impact of these increased costs on our national clean air policy, the EPA has supported a research and development effort aimed at evaluating existing novel devices and discovering and bringing to a commercial feasibility stage devices based on new collection principles or concepts or new combinations of existing concepts.

The Particulate Technology Branch of EPA's Industrial Environmental Research Laboratory, located at Research Triangle Park, North Carolina, for the past 5 years has had the major responsibility for carrying out the R&D effort mentioned above. This R&D program has been divided into the following program areas:

1. Characterization and Improvement of Conventional Control Devices.

2. Development of Technology for Control of Particulate Emissions from the Combustion of Low Sulfur Coal.

3. New Idea Identification and Evaluation (Novel Device Evaluation).

4. New Particulate Control Technology Development (Novel Concepts).

5. High Temperature and Pressure Particulate Control.

6. Accelerated Pilot Demonstrations.

This paper will discuss only program areas 3 and 4, above, associated with novel device and novel concepts research and development.

NOVEL DEVICE PROGRAM

The objective of IERL-RTP's novel device program is to identify and evaluate new technology or new combinations of well studied mechanisms in order to achieve cost effective control of fine particulate. A novel particulate collection device is a device or a dust collection system based on new collection principles or on radical redesign of conventional collectors which is available for testing as a pilot scale or full scale unit.

In the fall of 1973 the novel device evaluation program was initiated to identify, evaluate, and develop, where necessary, those devices or systems which showed the most promise for high efficiency collection of fine particulate. Contracts with four research organizations were funded to test and evaluate novel devices.

More than 40 novel particulate collectors have been identified. About half of the devices identified have been of sufficient interest to justify technical evaluations. To date 14 devices have been either field or laboratory tested:

Braxton - Sonic Agglomerator
Lone Star Steel - Steam-Hydro Scrubber
R. P. Industries - Dynactor Scrubber
Aronetics - Two-Phase Wet Scrubber
Purity Corporation - Pentapure Impinger
Entoleter - Centrifield Scrubber
Andersen 2000 - CHEAF
Rexnord - Granular Filter Bed
Air Pollution Systems - Electrostatic Scrubber (Scrub-E)
Air Pollution Systems - Electro-Tube
Century Industrial Products - FRP-100 Low Energy Wet Scrubber
American Precision Industries - Apitron
Particulate Control Systems - Electrified Bed
Ceilcote - Ionizing Wet Scrubber

Future plans include testing four additional devices if satisfactory test sites can be located:

Combustion Power - Dry Scrubber
United McGill - NAFCO ESP
Dart Industries - Hydro-Precipitrol Wetted Wall ESP
DuPont Company - DuPont Scrubber

The devices which have been tested by IERL-RTP's Particulate Technology Branch have not been selected for any particular industry. Although all were tested on sources with a high fraction of fine particulate, several were tested on metallurgical sources.

Novel Device Tests

This section discusses some of the more successful novel device tests with emphasis on those devices which may find application in the non-ferrous metals industry. Figure 1 presents a comparison of the fractional particulate penetration for most of the novel devices tested by EPA.

Of all the devices tested, the Lone Star Steel Steam-Hydro Scrubber gave the highest efficiency on fine particulate--99.9 percent efficiency on a fine fume from an open hearth steel furnace. This scrubber uses a high pressure steam nozzle surrounded by a number of water nozzles which inject water into the steam, generating very fine high velocity water droplets which are discharged into a venturi throat for capture of dust particles. A schematic of the system tested is given in Figure 2.[1]

The Steam-Hydro system provides 99 percent collection efficiency over a wide range of particle sizes. However, intrinsic power consumption is quite high, an order of magnitude greater than an ejector venturi scrubber. If fuel must be purchased to make the steam, the Steam-Hydro system is not an economic alternative to such systems as fabric filtration or electrostatic precipitation. If sufficient waste heat is available, the Steam-Hydro system becomes economically feasible. It should be noted that this system creates its own draft, thus obviating the need

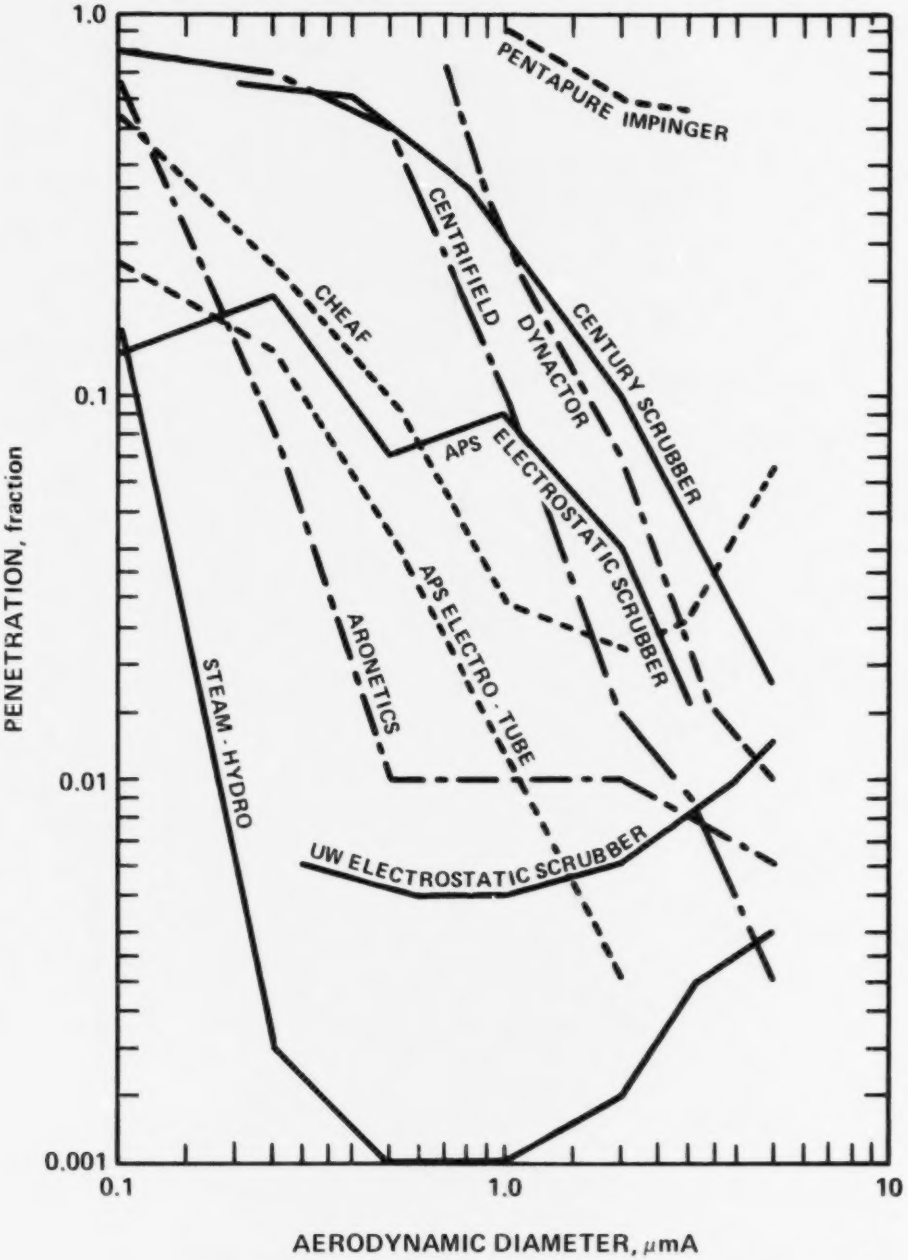

Figure 1. Penetration vs. particle size for novel devices tested by EPA.

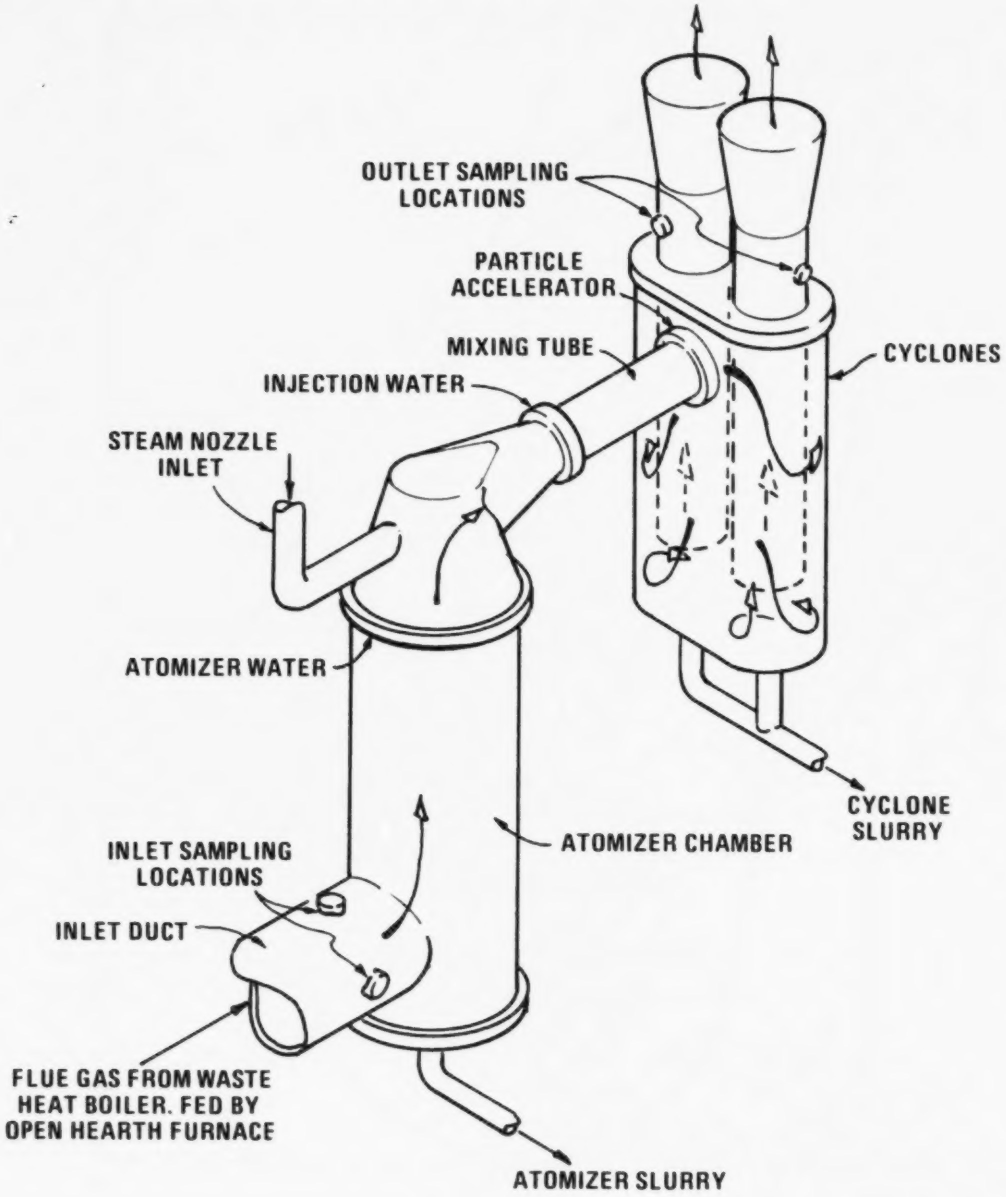
OUTLET SAMPLING LOCATIONS
PARTICLE ACCELERATOR
MIXING TUBE
CYCLONES
INJECTION WATER
STEAM NOZZLE INLET
ATOMIZER WATER
CYCLONE SLURRY
INLET SAMPLING LOCATIONS
ATOMIZER CHAMBER
INLET DUCT
FLUE GAS FROM WASTE HEAT BOILER. FED BY OPEN HEARTH FURNACE
ATOMIZER SLURRY

Figure 2. The Lone Star Steel steam-hydro air cleaning system.

for fans, blowers, etc. The Steam-Hydro system has been installed in open hearth furnaces, cupolas, iron ore kilns, paper mills, sinter plants, ammonium nitrate fertilizer plants, and copper smelters. Since the Steam-Hydro system was tested by EPA, Lone Star Steel has reported that they have made significant reductions in power requirements without loss of collection efficiency.

The Aronetics Scrubber is similar to the Steam-Hydro Scrubber but instead of steam it discharges high-pressure/high-temperature (200°C) water through a nozzle to provide small high velocity water droplets. The Aronetics unit, tested on a ferroalloy plant, was not quite as efficient as the Steam-Hydro unit. Collection efficiency was found to be greater than 50 percent for particles larger than 0.5 μm aerodynamic diameter, and greater than 99 percent for particles larger than about 1 μm aerodynamic diameter. A schematic of the system tested is given in Figure 3.[2]

The Steam-Hydro and Aronetics Scrubbers are very high energy users, but in both EPA tests waste energy was used to generate the steam or hot water.

In a field test on a diatomaceous earth calcining and drying process, the Cleanable (media) High Efficiency Air Filter (CHEAF) had an overall mass efficiency of 95 percent and maintained the efficiency down to about 0.3 μm. A laboratory test of this device confirmed that relatively high collection efficiencies can be maintained on submicron particulate. A schematic of the CHEAF System tested in the field is given in Figure 4.[3,4]

The manufacturer claims that the CHEAF system is well suited to the collection of water-soluble submicron particulates and aerosols, including ammonium nitrate and urea prill tower emissions, soda and borosilicate glass furnace emissions, phosphoric acid mists, phosphorus pentoxide fumes, emissions from inorganic chemical calciners and dryers, food product spray dryer emissions, galvanizing fumes, and sulfuric acid mists.

A small pilot scale laboratory test of the Air Pollution Systems (APS) electrostatic scrubber (Scrub-E) showed this system to be equal to a conventional venturi scrubber with a power requirement 1-1/2 to 2-1/2 times as great.[5] The Scrub-E is basically an electrostatic charger (or ionizer) followed by a venturi scrubber. Figure 5 is a schematic diagram of the pilot system. An electrode, upstream of the venturi, charges the inlet particles, which then enter the venturi throat. The gas stream atomizes the central water spray in the venturi throat. The charged particles, according to APS, are then attracted and collected by the highly polarized water molecules. The charged particles are also collected on the walls of the ionizer section prior to the throat of the venturi. A thin film of water is run down the inclined surfaces to keep the walls clear and prevent high voltage arcing. The particle laden water droplets are then collected by a cyclonic separator and sent into a settling tank (clarifier). Although the water can be recycled to the scrubber system,

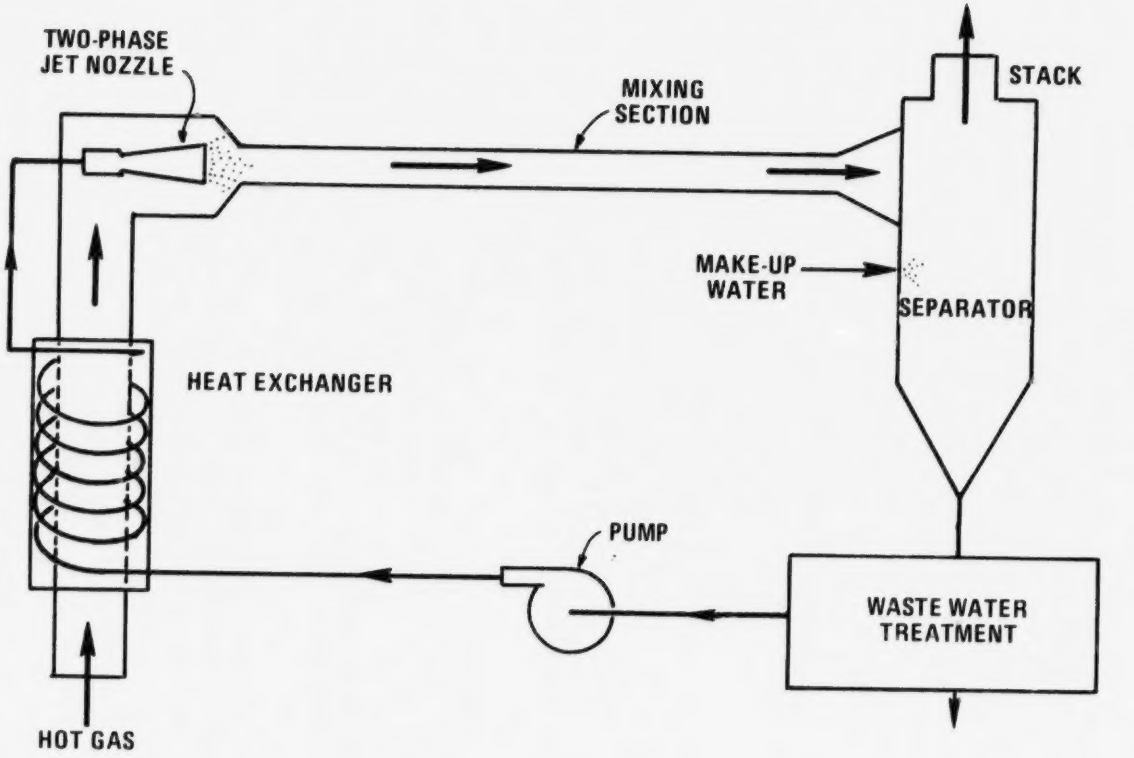
TWO-PHASE
JET NOZZLE
MIXING
SECTION
STACK
MAKE-UP
WATER
SEPARATOR
HEAT EXCHANGER
PUMP
WASTE WATER
TREATMENT
HOT GAS

Figure 3. Aronetics two-phase wet scrubber system.

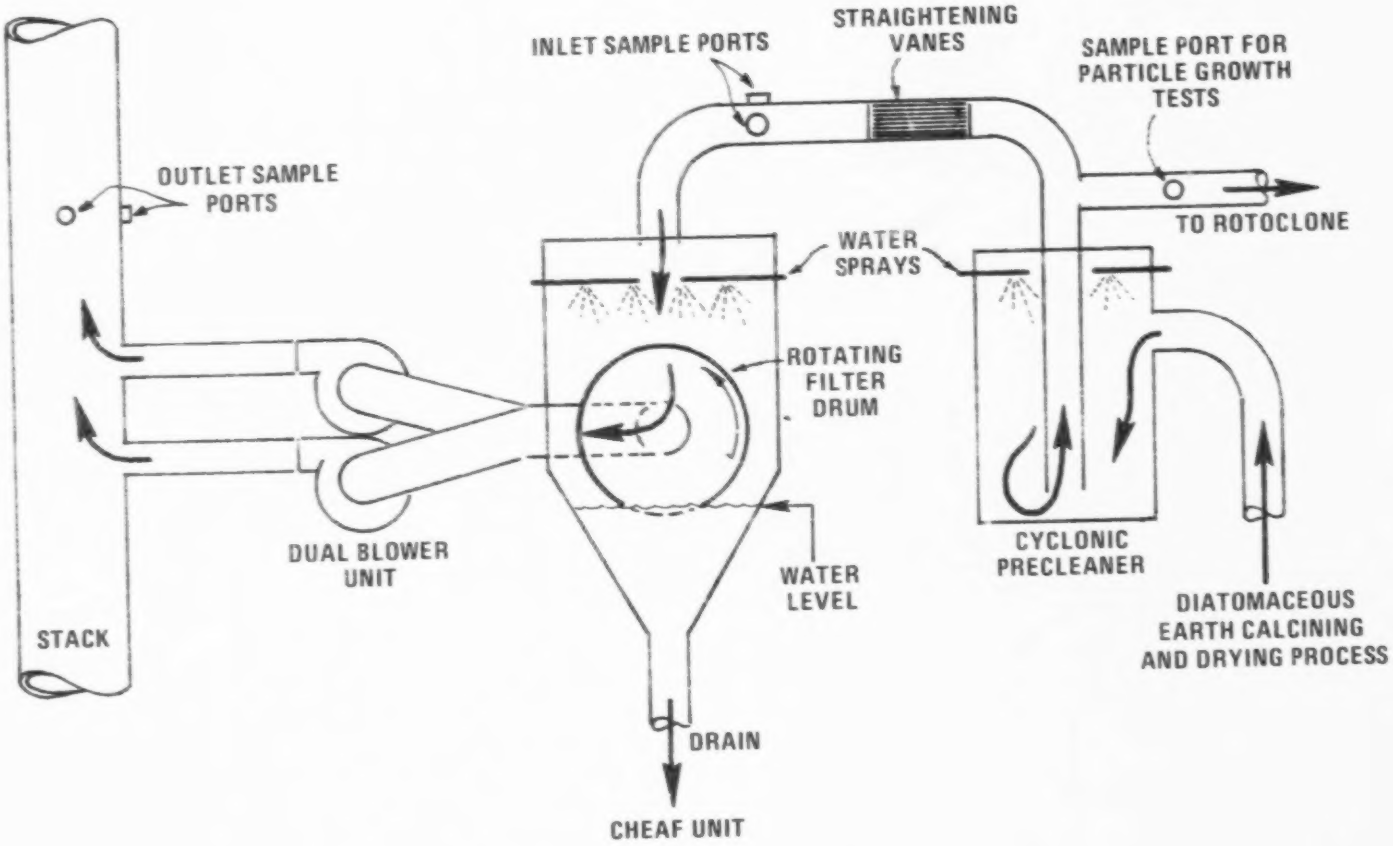
INLET SAMPLE PORTS
STRAIGHTENING VANES
SAMPLE PORT FOR PARTICLE GROWTH TESTS
OUTLET SAMPLE PORTS
TO ROTOCLONE
WATER SPRAYS
ROTATING FILTER DRUM
DUAL BLOWER UNIT
WATER LEVEL
CYCLONIC PRECLEANER
DIATOMACEOUS EARTH CALCINING AND DRYING PROCESS
STACK
DRAIN
CHEAF UNIT

Figure 4. Andersen 2000 CHEAF system.

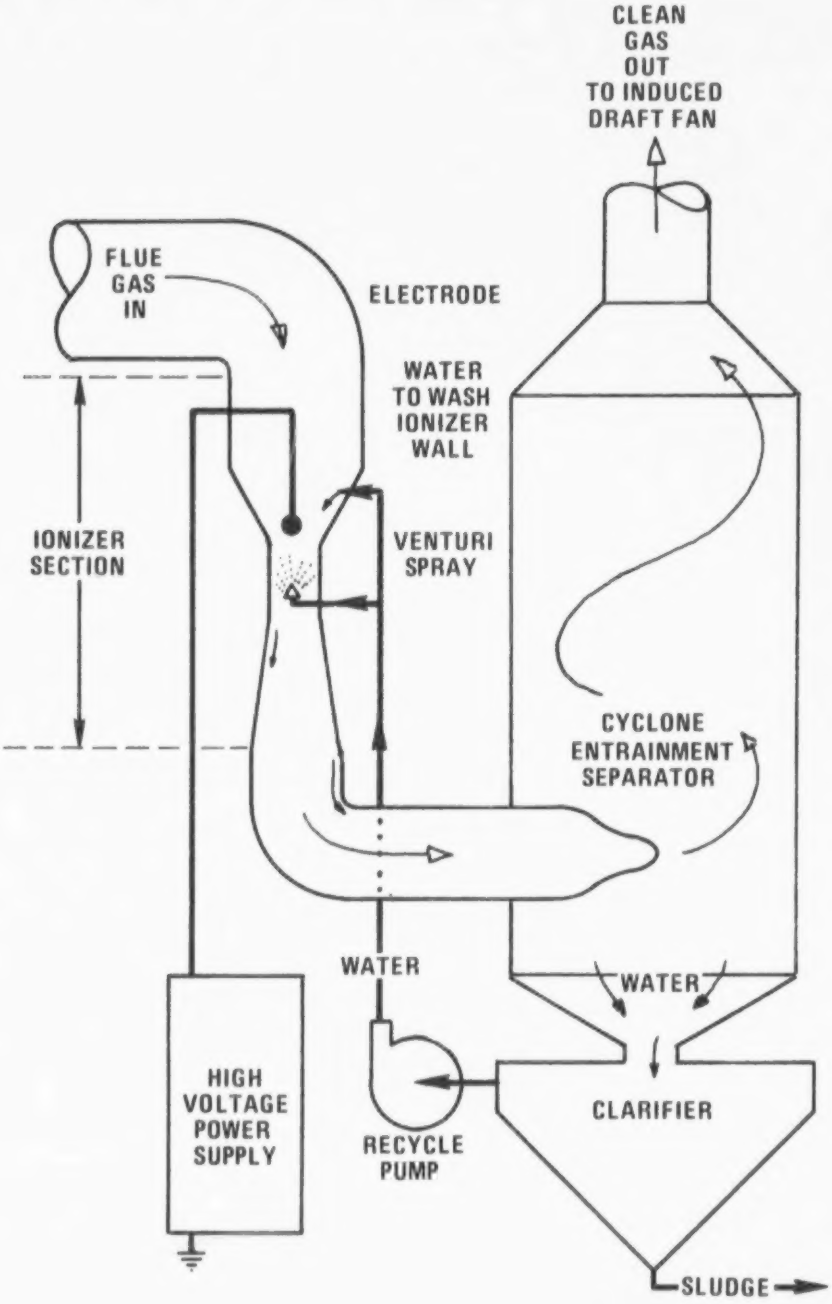
CLEAN
GAS
OUT
TO INDUCED
DRAFT FAN
FLUE
GAS
IN
ELECTRODE
WATER
TO WASH
IONIZER
WALL
IONIZER
SECTION
VENTURI
SPRAY
CYCLONE
ENTRAINMENT
SEPARATOR
WATER
WATER
HIGH
VOLTAGE
POWER
SUPPLY
RECYCLE
PUMP
CLARIFIER
SLUDGE

Figure 5. Air Pollution Systems Scrub-E.

fresh water was used during the test program.

The APS Electro-Tube, which is similar to a wet wall electrostatic precipitator, gave some very high efficiencies on fine particulates--as high as 98.9 percent on 0.5 µm particles.[6] This performance is similar to that which can be achieved in small wet electrostatic precipitators with the same ratio of plate area to volumetric flow rate.

The pilot scale APS Electro-Tube is basically a tube electrostatic precipitator with a central rod electrode and wetted wall collector. Figure 6 is a schematic diagram of the pilot system. The inlet particles are charged in a high energy field (12 kV/cm) by a high intensity ionizer at the base of the electrode. The charged particles then migrate to the wetted wall in the body of the device in a field of 5-10 kV/cm. APS indicates that an initial saturation charge on the particles, higher than the usual 4-5 kV/cm for a conventional ESP, facilitates increased migration in the collecting electric field.

The R.P. Industries' Dynactor Scrubber uses a proprietary nozzle design to produce a water spray which serves as an air mover and as a scrubber, thus cleaning and propelling the gas simultaneously. In an EPA sponsored laboratory test, the Dynactor Scrubber performed as a well designed scrubber but was found to have poor efficiency on fine particulate. A schematic of the system tested is given in Figure 7.[7]

In FY-77 a competitive procurement was issued to demonstrate at pilot or small full scale the technical and economic feasibility for the most promising existing novel particulate collection system for control of fine particulate emissions from industrial sources. This competitive procurement was won by Air Pollution Systems, Inc. (APS) for demonstration of the Scrub-E which was tested in the laboratory under the novel device evaluation program. A contract was funded in September 1977 with APS to demonstrate a 280 to 570 m^3/min (10,000 to 20,000 cfm) Scrub-E on a fine particulate source. A magnesium recovery furnace has been selected as the site for this demonstration. Startup is scheduled for April 1979.

NOVEL CONCEPTS PROGRAM

The major difference between the Novel Device and Novel Concepts program areas lies in the fact that the Device Program deals with existing equipment which either is offered commercially or is available for pilot scale testing, and the Concepts Program deals with the development of equipment from untried ideas or from the results of proven research studies which show that the development of an idea is feasible.

Particulate collection mechanisms utilized by scrubbers and fabric filters are impaction, interception, and diffusion and by ESP's are field and diffusion charging. This combination of mechanisms gives rise to a minimum in efficiency at the 0.2 to 0.5 µm range for conventional devices. Under optimum conditions, this minimum may be greater than

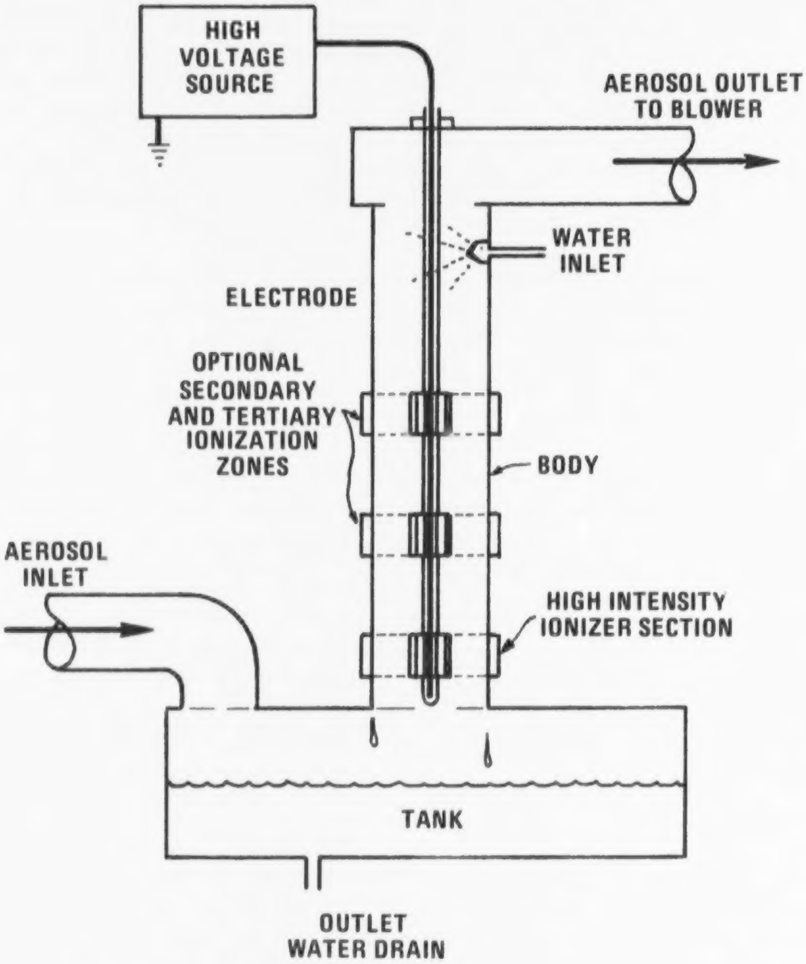
HIGH VOLTAGE SOURCE
AEROSOL OUTLET TO BLOWER
WATER INLET
ELECTRODE
OPTIONAL SECONDARY AND TERTIARY IONIZATION ZONES
BODY
AEROSOL INLET
HIGH INTENSITY IONIZER SECTION
TANK
OUTLET WATER DRAIN

Figure 6. Air Pollution Systems Electro-Tube.

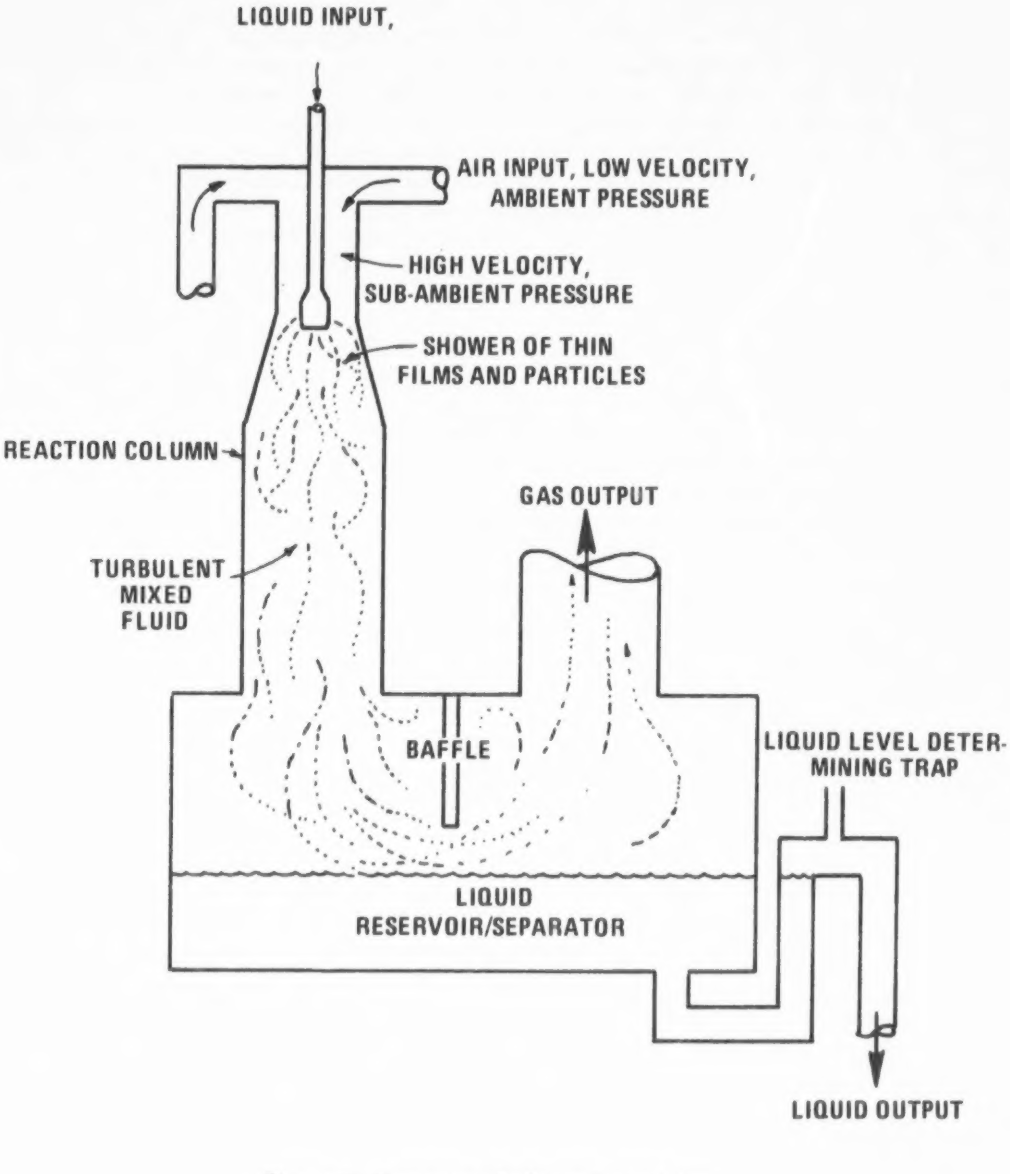
LIQUID INPUT,
AIR INPUT, LOW VELOCITY, AMBIENT PRESSURE
HIGH VELOCITY, SUB-AMBIENT PRESSURE
SHOWER OF THIN FILMS AND PARTICLES
REACTION COLUMN
GAS OUTPUT
TURBULENT MIXED FLUID
BAFFLE
LIQUID LEVEL DETERMINING TRAP
LIQUID RESERVOIR/SEPARATOR
LIQUID OUTPUT

Figure 7. Dynactor single-stage scrubber.

90% for ESPs, and greater than 99 percent for fabric filters. However, under other conditions (e.g., high temperature, high ash resistivity, sticky particulate, and corrosive or explosive flue gases), new concepts specific to a problem will have an advantage.

Most work to date has been directed toward combining electrostatic removal mechanisms with scrubbing or filtration mechanisms. The first area to be developed was charged droplet scrubbing, with a feasibility study at M.I.T. and a pilot demonstration at TRW on a Kaiser coke oven.[8] Electrostatics and filtration have been studied at both Battelle Northwest and Carnegie-Mellon: the former with bed filters; the latter with baghouses. At least two new concepts, a ceramic membrane filter and a magnetic fiber bed, are oriented toward cleanup of high temperature gases (1000-2000°F/500-1100°C). Other new concepts which have been studied are foam scrubbing and pleated cartridge filters of a novel material.

Of the advanced electrostatic collection concepts studied, those employing water droplets or filters have demonstrated enhanced performance and should be considered for future applications. Electrostatic collection with water drops shows high removal efficiencies for 0.5 μm particles which are difficult to capture. Electrostatic collection with filters shows the potential for operation at either lower pressure drops or higher filtration rates. The conventional approach in using electrostatics for abatement of airborne particle emissions is to charge the particles and then collect them in a high voltage DC field. The particles can be both charged and collected in a single device. Alternatives to this approach involve precharging the particles in one device, and collecting them in another. The collection device may use electrostatic forces or may introduce a collecting medium such as water droplets or a filter.

Some electrostatic collection concepts are given in Table 1. Details for these concepts are given below. In general, the entries in Table 1 reflect the three major categories of collection mechanisms: electric field effects, scrubbing, and filtration. For the last two, combining electrostatic effects and conventional mechanisms enhances performance over conventional devices of the same type. The possibility of enhanced performance stimulated EPA's involvement in developing advanced electrostatic collection concepts.

Electrostatic Collection with Droplets

The use of water droplets in an electrostatic collection device will differ according to whether the drops are charged or not and whether there is an ambient electric field. The usual configuration for a charged droplet scrubber is an ambient field with the drops charged oppositely to the particles. The scrubber consists of three chambers: (1) a corona discharge section for particle charging; (2) a spray chamber which introduces oppositely charged droplets; and (3) a mist eliminator. The University of Washington electrostatic scrubber[9] is a charged droplet scrubber tested by EPA. The scrubber, shown schematically in Figure 8, was tested on a coal fired boiler side stream at 1655 to 1802 am^3/hr

Table 1. ELECTROSTATIC COLLECTION CONCEPTS

Name	Collection by Means of	EPA Activity
Electrostatic Scrubbing	Electric field and water droplets	Contract 68-02-0250, MIT Contract 68-02-1345, TRW
Charged Droplet Scrubbing	Water droplets	Grant 803278, U. of Washington Grant 804393, U. of Washington
Electrostatic Fiber Beds	Filter fibers	Grant 801581, Battelle Northwest
Electrostatic Effects in Fabric Filters	Filter fabric	Grant 803020, Carnegie-Mellon U.

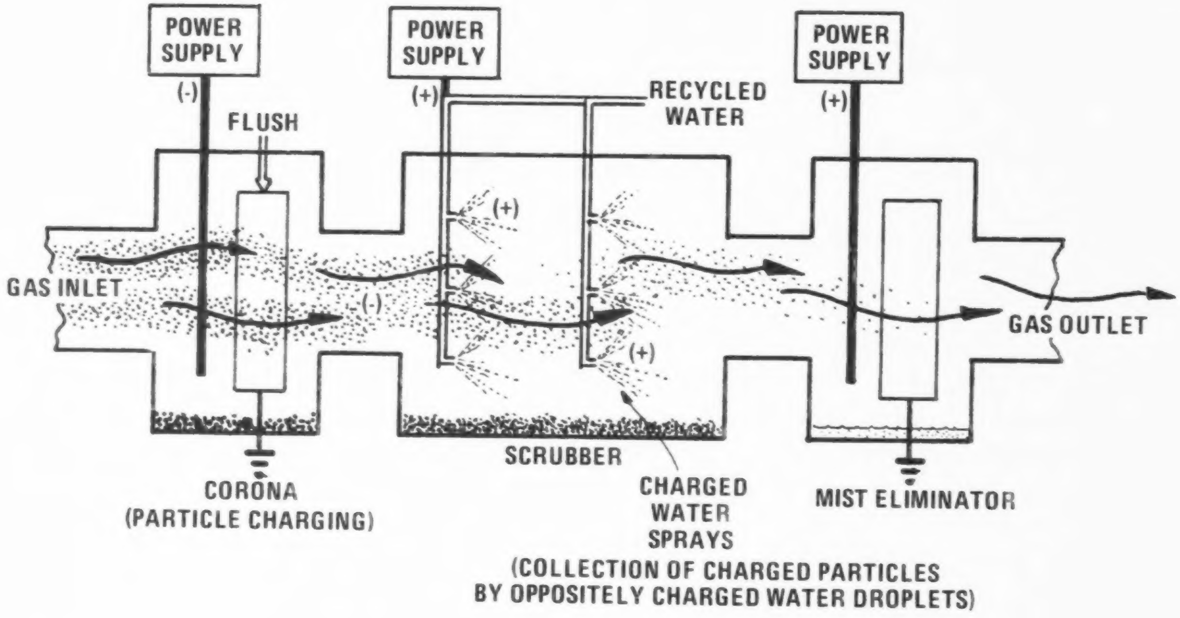
POWER SUPPLY
(-)
FLUSH
POWER SUPPLY
(+)
RECYCLED WATER
POWER SUPPLY
(+)
(+)
(-)
(+)
GAS INLET
GAS OUTLET
SCRUBBER
CORONA (PARTICLE CHARGING)
CHARGED WATER SPRAYS
(COLLECTION OF CHARGED PARTICLES BY OPPOSITELY CHARGED WATER DROPLETS)
MIST ELIMINATOR

Figure 8. University of Washington electrostatic scrubber.

and on an electric arc steel furnace side stream at 1754 to 3031 am^3/hr. In both cases the overall efficiency was greater than 96 percent; performance of the scrubber was greatly influenced by the water to gas ratio. At the coal fired boiler site, doubling the water to gas ratio from 0.32 to 0.76 liters/am^3 halved the overall penetration from 3.9 to 1.9 percent. The power consumption was approximately 13 W/(m^3/min) and the residence time was 8 sec.

A second case, which does not use a precharging section as other devices in this group do, has an ambient field and the charge on the drops is imposed by the electric field. This device, referred to as a charged droplet precipitator, has only one chamber: water is introduced into its middle from nozzles at high voltage. The drops leave the nozzles with a charge and are accelerated to the walls by the electric field. The TRW charged droplet scrubber, a charged droplet precipitator tested by EPA, was demonstrated on a coking oven battery exhaust.[10] The test unit, shown schematically in Figure 9, has a capacity of 31,000 m^3/hr at a gas flow rate of 1.83 m/s with a residence time of 3 sec. The scrubber operated with a low water rate of less than 0.14 liters/am^3 (1 gal./1000 acf). The total power consumption was 28-42 W/(m^3/min). The demonstration testing showed that the TRW unit could be operated with very infrequent wall wash and that the overall efficiency (approximately 94 percent) was highest under this condition.

A comparison of advanced electrostatic concepts using water droplet collectors is given in Table 2. These results are from development programs, as noted in the introduction, or from field tests and may have been tested under the Novel Device Program. All concepts provide high overall collection efficiencies and high efficiencies in removing the 0.5 μm diameter particle. It should be noted that penetrations range from 30 to 50 percent for 0.5 μm particles in conventional scrubbers.[11] Since penetrations for the 0.5 μm particle through the advanced electrostatic collection devices range from 1 to 10 percent, these concepts have demonstrated significantly enhanced performance. It is possible to increase the pressure drop in a venturi scrubber to decrease the penetration of 0.5 μm particles to 10 percent. The pressure drop required can be estimated by Calvert's equation 5.3.6-12 in the Scrubber Handbook.[12] The energy requirement for a venturi scrubber to collect 90 percent at a 0.5 μm particle size is 325 W/(m^3/min). This is an order of magnitude higher than the 13-80 W(m^3/min) reported for the same performance with electrostatic droplet concepts.

Electrostatic Collection with Filters

The use of filters with electrostatics varies according to the collector (i.e., fibers or a fabric). For a fiber collector, precharging the particles enhances collection efficiency. For a fabric collector, precharging the particles changes the nature of the deposited cake, correspondingly lowers the pressure drop, and maybe improves collection efficiency.

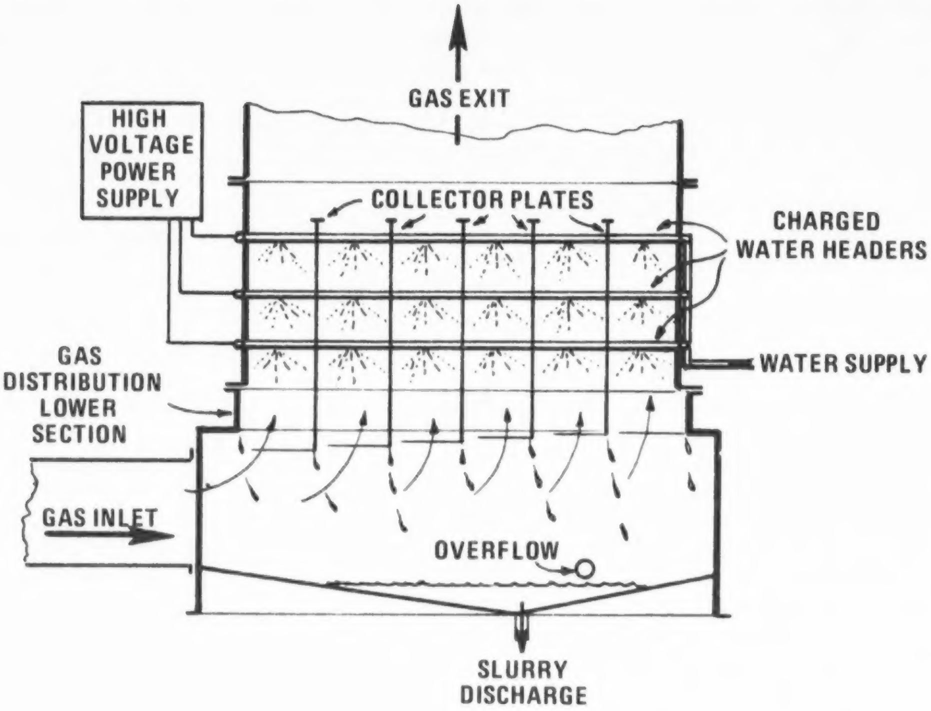
GAS EXIT
HIGH VOLTAGE POWER SUPPLY
COLLECTOR PLATES
CHARGED WATER HEADERS
WATER SUPPLY
GAS DISTRIBUTION LOWER SECTION
GAS INLET
OVERFLOW
SLURRY DISCHARGE

Figure 9. TRW charged droplet scrubber.

Table 2. COMPARISON OF ELECTROSTATIC DROPLET CONCEPTS/DEVICES

Unit Tested	Overall Efficiency, %	Power Requirement W/(m^3/min)	Water to Gas Ratio (l/Am^3)	Percent Efficiency at	
				0.5 μm	1.0 μm
U. of Washington[a] Electrostatic Scrubber	98.1-99.5	13	0.76-0.77	99	97.5
U. of Washington[b] Electrostatic Scrubber	96.4-98.6	13	2.23-2.29	95	97
APS Electrostatic Scrubber	83-97	80	1.44	96	90
TRW Charged Droplet Scrubber	93.5	28-42	0.11-013	90	85-95

[a] At a coal fired boiler.

[b] At an electric arc furnace.

The first case, referred to as an electrostatic fiber bed, was studied by Battelle Northwest.[13] The three aerosols studied (NH_4Cl, Na_2O, and MgO) had mass median diameters of less than 1 μm. The freshly generated particles were drawn first through a corona charging section and then through the fiber bed. The fiber bed had a void fraction of 0.96, was 15 or 30 cm thick, and was composed of stainless steel, polypropylene, or Teflon. With a 30 cm bed, collection efficiency was greater than 95 percent for bed velocities up to 1 m/s with a pressure drop through the clean bed of less than 1 cm WC.

The second case, precharging particles before a fabric filter or electrostatically augmented filter, was studied by Carnegie-Mellon University.[14] The filter was a 10 cm ID by 30 cm long bag made of woven material such as polypropylene without antistatic treatment. The aerosol was a silica dust charged either by impingement against a tungsten carbide surface or by corona discharge. The bags were pulse-jet cleaned. At an air-to-cloth ratio of 6, the electrostatically augmented filter had a pressure drop of 6.4 cm WC; a comparable bag without precharged particles had a pressure drop of 16.5 cm WC. Similar results have been reported by American Precision Industries, Inc.[15] Testing their electrostatically augmented filter, call the APITRON, American Precision Industries reports a reduction in pressure drop from 10 cm WC for a conventional filter to approximately 1 cm WC for the APITRON, maintaining the same filtration rate on a steel furnace fume. If the pressure drop is the same for both filter types, the APITRON may be operated at a filtration rate 4 times greater. Since electrostatically augmented filters have demonstrated significant reductions in pressure drop at the same filtration rate (or significant improvements in filtration rate at the same pressure drop in conventional filters), this concept also has proven enhanced performance with electrostatics.

High Gradient Magnetic Collection

Another collection concept which has been developed by EPA/IERL-RTP's Particulate Control Program uses a magnetic force as the prime collection mechanism. A device based on this new technology, magnetic filtration, has been developed. We call it high gradient magnetic separation (HGMS).

Recently HGMS has been used successfully to purify various types of wastewater.[16] Conventional magnetic separation devices are generally useful for separating relatively large particles of strongly magnetic materials such as iron and magnetite. HGMS, by contrast, makes possible the efficient separation of micron-size particles of weakly paramagnetic materials from liquids at high process rates. Generalized theory indicates that HGMS can also be used to remove particles from gas streams. Magnetic forces can be cost effective if the control device is properly designed.

In its simplest practical form, the HGMS is a canister packed with ferromagnetic fibers (such as a stainless steel wool) that are magnetized by a strong external magnetic field (Figure 10). The strong magnetic forces produced near the edges of the fibers are capable of trapping

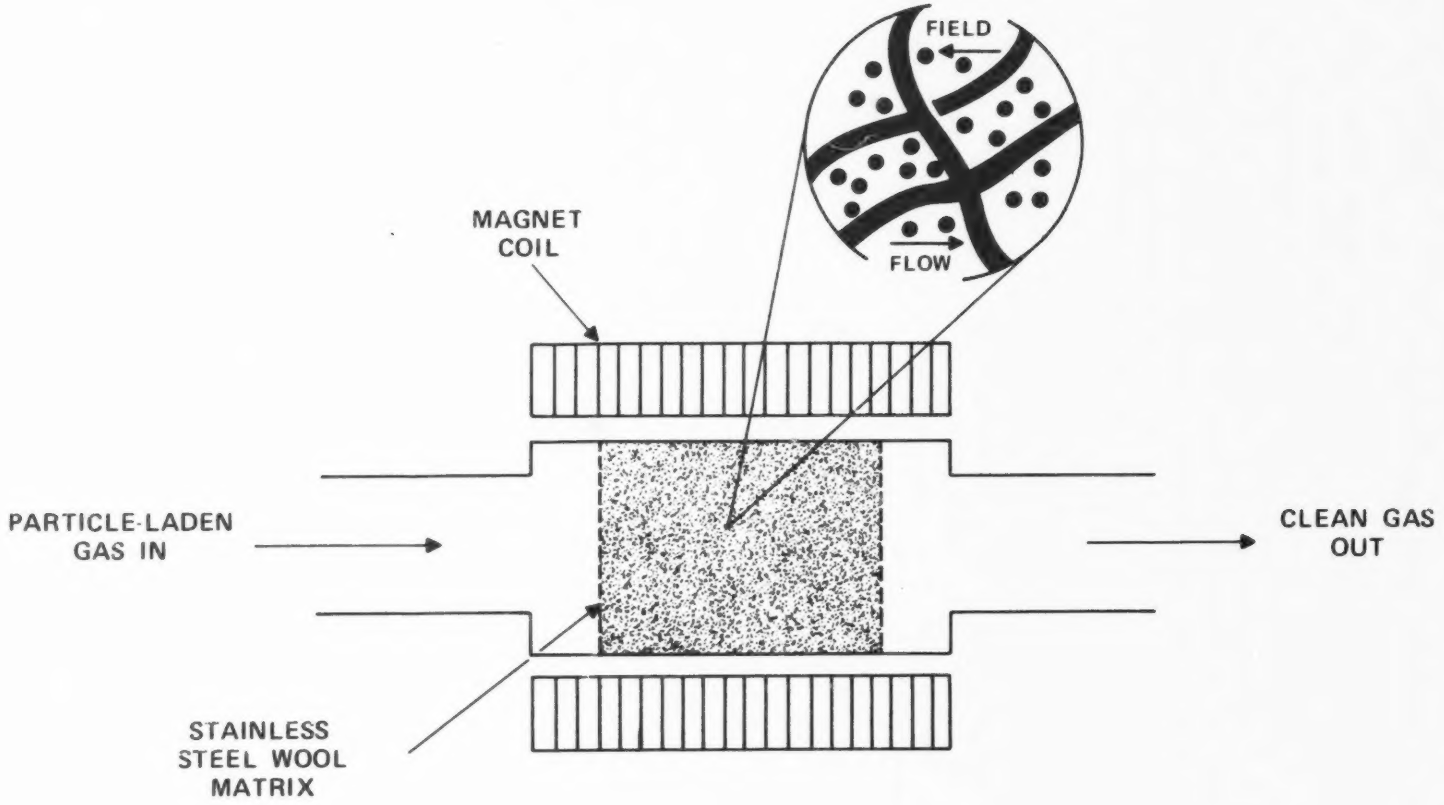
FIELD
FLOW
MAGNET COIL
PARTICLE-LADEN GAS IN
CLEAN GAS OUT
STAINLESS STEEL WOOL MATRIX

Figure 10. Schematic representation of a high gradient magnetic separator.

fine, weakly magnetic particles with high efficiency.

The packing density of the separation zone can be adjusted to obtain a proper balance between conventional filtration phenomena and the HGMS effect. Reducing the packing density lowers the pressure drop through the separator but requires a higher magnetic field for efficient particle collection. The overall particle collection efficiency is theoretically a function of the applied magnetic field, of filter mesh parameters (fiber diameter and magnetization, packing density, and length of mesh in the direction of flow), of particle parameters (magnetic susceptibility and diameter), and of fluid parameters (superficial velocity and viscosity).[17]

This concept has been proven to be feasible for the collection of fine dusts which are weakly magnetic in nature. Laboratory pilot scale experiments have been completed using BOF dust as a model (see Figure 11), and the construction of a portable pilot unit is now underway. This unit is scheduled for its first field test on a gas stream from an iron and steel sinter plant in 1979.

CONCLUSIONS

The feasibility of enhancing the collection of fine particulate by conventional devices utilizing electrostatic techniques has been demonstrated. In addition, devices utilizing new collection mechanisms or new combinations of old mechanisms have shown promise and are currently being developed. The economics of these new approaches have not yet been established.

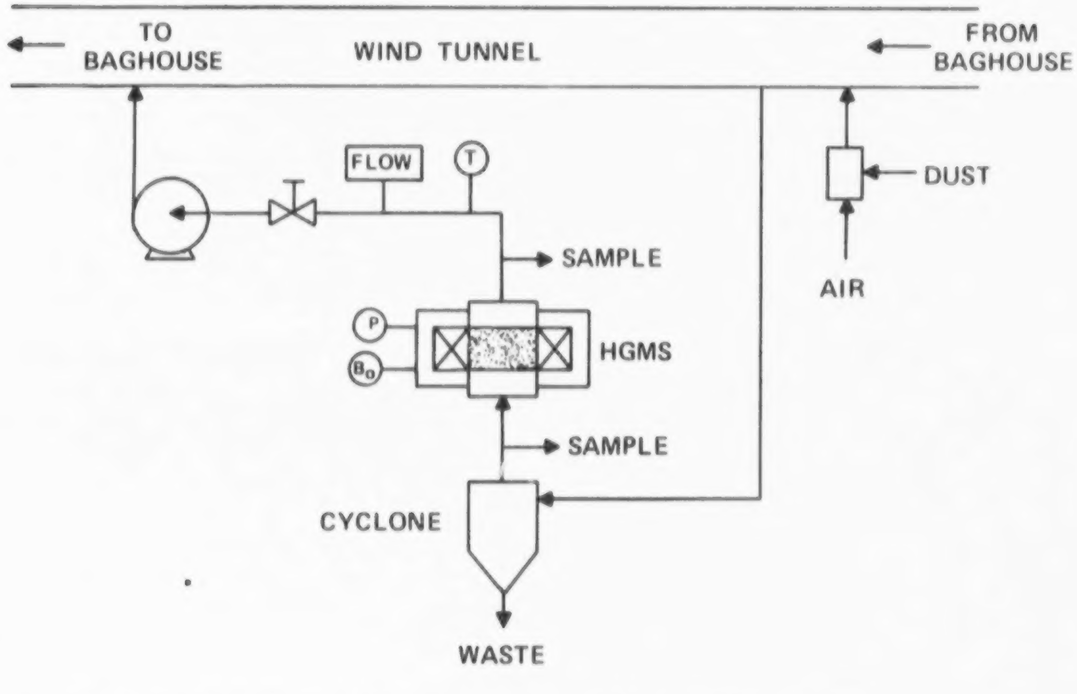
TO
BAGHOUSE
WIND TUNNEL
FROM
BAGHOUSE
FLOW
T
DUST
SAMPLE
AIR
P
HGMS
B_0
SAMPLE
CYCLONE
WASTE

Figure 11. Diagram of bench-scale apparatus used in preliminary HGMS experiments.

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SYMPOSIUM ADDRESS

"And What of the Future?"

W. H. Dresher
Dean, College of Mines
The University of Arizona
Tucson, Arizona

AND WHAT OF THE FUTURE?

W. H. Dresher
Dean, College of Mines
The University of Arizona
Tucson, Arizona

It is a distinct honor and a privilege for me to have been invited here to address your conference tonight. When Dr. Oglesby invited me, I accepted with great haste thinking that this would be one more opportunity for me to practice my favorite pastime of complaining about what the Federal Government is doing wrong today, when he interrupted me to say..."but the Environmental Protection Agency is sponsoring the conference". My bubble of euphoria burst immeidately when I had to agree with what he was implying, that it would be extremely rude for an invited guest to insult a host by throwing bricks at him. I am here tonight, so obviously I accepted the invitation and promised to be on my best behavior. But more than that, I accepted a challenge to cause myself to think and to bring to you tonight a message which, in turn, hopefully will cause you to think. Specifically, where is the road we are taking in American society today leading us? Will the actions which we are taking truly enable us to arrive at where we think we are going? All too often today our actions are taken in the name of the future and the well-being of future generations of Americans. I have to ask myself, and tonight I will ask you to ask yourselves, "and what of the future?" In saying this, I am reminded of a bumper sticker I once saw which said, "Due to uncertainties today, tomorrow is cancelled." Will our tomorrow be cancelled due to the uncertainties of the actions we take today? Oh yes, tomorrow will come all right, but will it hold for us the opportunities and the qualities which we expect?

I have become intrigued recently to try to satisfy myself with an understanding of just what we are all about today. We have come a long way as a country and as a culture and we of the present generation have seen more human progress on many fronts than any other generation of people. Some of us have seen the whole range of technological development from the horseless carriage to space travel. We have seen a virtual explosion of communications and information systems. We have seen once fatal and disabling diseases all but eliminated and we have seen a

Green Revolution provide a bountiful harvest where none was possible before. But, on the other hand, we have seen some very disturbing things, too. We have seen the race riots of the 1950's and the 1960's. We have seen some of our most capable leaders (including a president) gunned down in cold blood. We have seen urban problems come to a head that no amount of public money has been able to solve. More in keeping with the theme of this conference, we have seen an environmental concern grow to a magnitude that would have been totally unbelievable twenty years ago if a modern day Nostradamus had forecasted the future. To top it all off, we have seen the development of an energy crisis which promises to complicate any goals which we might set for the future.

There is no doubt that we are in a period of change, but it is difficult to understand the nature of the change which is upon us. In a generation which should be, and probably is, better off than any previous generation in history, we are beset with problems and faults. As Pogo has often been quoted as saying, "...we have met the enemy and they are us." Perhaps it is because of our prosperity that we have the time and the luxury to find fault with ourselves. I would like to think that this is so, for it gives me the ability to rationalize, and perhaps to begin to understand, what is happening to us. If this is possible, I believe we should be able to plan our route into the future. If it is not possible, then we are like Dorothy in the Wizard of Oz....we may find the yellow brick road and the Wizard at the end of our traumatic travels, but then again we may land in a pile of fragments and splinters in the middle of a Kansas wheat field, our society never to rise again to its former stature in the world.

About a decade ago, after the Korean War and the race riots of the fifties, and the Vietnam action and the youth rebellion of the sixties, we saw a number of people attempt to deduce and to articulate what was happening to our society. Alvin Toffler's "Future Shock" and Charles Reich's "Greening of America" are two notable examples of these attempts. As Reich put it,

> "There is a revolution coming. It will not be like revolutions of the past. It will originate with the individual and with culture, and it will change the political structure only as its final act. It will not require violence to succeed, and it cannot be successfully resisted by violence. This is the revolution of the new generation."

"Revolution" seems to be the key word in most people's interpretation of what we are about. John D. Rockefeller III, in his book, "The Second American Revolution", describes the steps which our society is taking to evolve from "materialism" to "humanism". He points to the Declaration of Independence and the Constitution of the United States as the basic credo of the revolutionary change which he sees in progress. According to Mr. Rockefeller,

> "The Second American Revolution--the humanistic revolution --is one of fulfillment, designed to bring finally to fruition in modern times not only the letter, but the full spirit, the intent of these great documents. To grasp this, it seems to me (Mr. Rockefeller), is essential to understand what is going on today."

I like to think that I understand what these people are telling me. I like to think that I might even believe in the cause which is being espoused. However, it scares me to think of how these concepts can be misused: how overly-dedicated zealots can demand utopia now, at a rate of change that we cannot accommodate; how the unscrupulous might use this period of amorphous behavior of our society to impose their will of yet another society upon us. Perhaps this society will be the one which former Secretary of the Treasury, William Simon, describes in his book, "A Time for Truth"--one in which individual freedom is lost once our industrial civilization collapses and "humanism" gives way to panic, chaos, riots, starvation, and death. I share this fear for the future with Secretary Simon.

For want of a better description, many futurists have termed the era which we are approaching as the "Post-industrial Society". Unfortunately, this terminology has two meanings and unless our status is precisely described and understood, the term, post-industrial society, can lead us to trouble. This, I firmly believe is part of our problem today. The first meaning, and in my opinion the wrong meaning, stems from the concept that industry is bad, it is the source of all of our problems today, it must be eliminated, and hence our society will certainly be a post-industrial society. I firmly believe that many of the radicals and zealots which we see using the environmental movement as their rallying point espouse this interpretation of the post-industrial society and use their stand for environmental protection as the vehicle to destroy the industrial structure of our country. These are the people who are likely to lead us to the society which Bill Simon fears. The other interpretation of the "Post-industrial Society" can be likened to the period in our history when we evolved from an agrarian society to an industrial society. In this process we did not destroy our farms for we still required food for sustenance. The evolution from one status to the other merely meant that we no longer had to focus a majority of our efforts on growing food. With the development of machinery, less and less of our populace was required to provide our food needs. In today's language, agriculture was becoming less and less a percentage of our gross national product as manufacturing took up a larger and larger scale. Similarly today, our transition is marked by the manufacturing component declining in our Gross National Product, with the service component expanding beyond 50 percent of our national efforts. Machinery now allows us more leisure time and more freedom from working, merely to survive. Machinery, if you will, gives us the time and the resources to seek humanistic goals.

Probably the clearest explanation of the phenomena which we are experiencing today is that proposed by Abraham Maslow of Brandeis University in 1954 (which was far too early to have predicted our status today). Maslow's "Hierarchy of Human Need" of "Theory of Self-Actualization", in my opinion, holds the key to what we are observing today and, consequently, holds the answer to what we must do to preserve and accomplish our goals for the future. According to Maslow, mankind aspires to a successive series of needs and/or desires in his quest for total fulfillment. First come physiological needs, such as food and shelter. Once these needs are satisfied, safety needs become the primary focus followed by social needs, ego needs, and finally self-fulfillment needs. Dr. Maslow proposes that a satisfied need is no longer a motivator and therefore tends to be forgotten; however, he also points out that a once-fulfilled need, if removed, causes a reversion to the lower level of concern. Thus, based on Dr. Maslow's theory, a post-industrial society cannot truly exist, by its creation, the industrial society upon which it is based, is destroyed. If it is true that our society is moving forward to a new higher level of human existence, then a new question arises. Will society be wise enough to make this transition in an uninterrupted manner or will it be caused to fall back to a lower level due to the failure of its economic system? Many believe that an adequacy of raw material and energy supply are at the heart of the answer to this question. I tend to be of this persuasion. Let's look now at some of the details of my concerns and relate these concerns to the subject of this conference, pollution control in the nonferrous metals industry.

In the last fifteen or so years, many Americans seem to have concluded that industry, particularly the basic industries, cannot or will not undertake change for social benefit without the imposition of government authority. The net result has been the promulgation and the proliferation of a literal niagara of federal regulations. None of these mandates individually has been cognizant of the relative costs and benefits to society, or (more importantly) of the aggregate influence of the total regulatory body. One recent study has determined that the copper industry, and indeed most nonferrous metals industries, is currently subject to more than 130 major regulatory mandates incorporating several thousand individual regulations. I have no doubt that each of these pieces of legislation, when originally derived, was designed to correct a real or perceived malfunction in industry's relationship to our society.

Unfortunately, as we are now learning, the cost of the resulting body of regulation (which, incidentally must be borne by society sooner or later) was ill-defined at the time of passage of the legislation. Only today are we becoming aware of the high cost of these regulations as our inflation spirals upward. The President's establishment of the Regulatory Council, with EPA's own Douglas Costle as its chairman, is the first step in attempting to take a positive step in the direction of reducing the regulatory cost impact on our society. It would seem that we have reached that unenviable state of requiring regulators to regulate regulators. The NRDC vs. Schultze case has underlined this absurdity by implying that the President, through his Council of Economic Advisors, has no business interfering in the regulatory activities of the agencies under his control. In spite of this evidence of concern on the part of the President, I cannot help but wonder if the full impact of the regulatory morass will be brought to light by the Council. Let's look at what I mean by this comment.

As I postulated earlier, the key to the success of our society in reaching the goals it appears to have established for itself has to lie in the preservation of the industrial base which supports it. And yet, as I overview today's happenings, I see one impediment after another being erected to undermine the stability of our industrial base. The EPA and the CEQ, in their recently released report, "Microeconomic Impact of Federal Pollution Control Programs: 1978 Assessment", gives cold, lifeless statistics. Wholesale prices will increase 0.4 percent per year, consumer prices 0.3 percent per year and productivity will decline 0.3 percent per year--all as a result of federal pollution control regulations. Others, such as Chase Manhattan Bank, tell us that the cost of government regulation to United States society in 1977 was $100 billion--almost $500 for every man, woman, and child that year. What these numbers do not tell us is where we will obtain the raw materials and energy to operate our society if we destroy our basic industries. Because of federal environmental, health, and safety regulations, two things are happening: 1) we are rapidly destroying our ability to obtain the vital fuel and nonfuel minerals from domestic resources which are required to feed our industry, and 2) we are rapidly destroying our ability to process these minerals into important raw materials. In both instances we are rapidly increasing our dependence on the vagaries of foreign governments for the fulfillment of our basic raw materials and energy needs. The energy scenario is rapidly unfolding and its consequences are already becoming evident. The nonferrous metals scenario is just beginning to develop and its direction is already just as evident...we are literally exporting our basic industries to foreign shores where foreign influences can control our future!

Between 1969 and 1975 eight United States zinc smelters closed down in Oklahoma, Montana, West Virginia, Illinois, and Texas. During this same decade, imports of zinc ores and concentrates dropped 77 percent while imports of zinc metal increased 89 percent. The net result has been that United States manufacturers are placing a far greater reliance on imported processed zinc than in the past. Zinc, I will point out, is vital to make automobiles and appliances more resistant to corrosion in order that they may last longer--one of the major goals of our new "conservation ethic".

For lead, the effect of regulation is not yet so obvious as for zinc. We are less reliant on foreign sources for lead than we were a decade ago. However, this statistic does not reveal the problem for lead. Recently adopted OSHA standards of 50 micrograms of lead per cubic meter of air based on an 8-hour time weighted average at the work place promise to force secondary lead smelters out of business. OSHA has recently estimated that the newly-imposed EPA and OSHA standards for the allowed concentration of lead in the general environment and the work place will require the expenditure of $1 billion and an operating cost of $300 million per year. By comparison, the annual product of the entire domestic lead industry is worth only $800 million! As a result, NL Industries, the largest producer of primary and secondary lead, is rapidly divesting itself of its lead operations, leaving the business of lead recycling to the smaller operators who can ill afford the costs of conforming to the regulations. Lead will be required in even greater quantities if electricity-powered vehicles are to help to relieve our petroleum deficiency or if solar and wind energy are to supplement fossil fuels. Lead is the only metal for which today's needs are satisfied by more than 50 percent recycled material. An average of 80 percent of the lead used in storage batteries is recycled lead. Again, recycling is another of our "conservation ethic" goals.

In copper, the capital outlay already invested in pollution abatement facilities has moved the industry to its limit of capital indebtedness. The long-term debt of the domestic copper industry today is 30 to 40 percent of its net worth. An original Federal government estimate in 1969 of a capital cost of $33.8 million to $81.0 million to remove 98.8 percent of the sulfur dioxide effluent from the entire nonferrous metals industry has already cost the copper industry alone in excess of $1.1 billion and 90 percent abatement has not yet been reached. Department of Commerce estimates are that the industry will spend $4.5 billion by 1987 in capital and operating costs and will be forced to reduce production by 36 percent relative to 1974 production in order to meet present EPA and OSHA requirements. According to this estimate, by 1987 our dependence on foreign copper will increase to almost 46 percent of our needs compared with an average of less than 10 percent over the last 20 years. Our present smelter capacity is just below our domestic consumption. Newly-imposed OSHA regulations on arsenic in the work place environment promise to close down at least one, and possibly two, domestic copper smelters. United States reliance on foreign sources has already increased from 6 to 8 percent of consumption in the early 1970's to almost 20 percent in 1978. Copper is a vital material in the generation, transmission, and conversion of electricity and will be vital to the substitution of electricity for natural gas in household use. This should be one of the major goals of the country if we are to reduce our import dependence for fuels.

The aluminum industry has recently revealed that the uncertainties of the nation's energy policy and the environmental obstacles to expansion have caused it to look outside the United States (especially to Brazil and Australia) to place the major share of its future investment. The immediate projection is for a shortage of aluminum and a drastic increase in prices. Ironically, the industry predicts that it may not be able to meet the needs of the automobile industry as it attempts to meet 1985 mileage standards by increasing the amount of aluminum in an automobile. Again, a conflict with a major goal of the United States in reducing its import dependence for petroleum.

Jordan Baruch, Assistant Secretary of Commerce for Science and Technology, stated the problem well in a speech recently when he said, "The whole body of regulation is primarily an ethic to stop industry from doing things perceived as against the public interest. Regulation is a tool for stopping industry. Regulation is negative in concept. It's like brakes on a car--essential but negative." It seems to me that it is most imperative that we find ways of preventing these brakes from locking. Regulations cannot be permitted to "stop" any industry so vital to the very existence of our society as our nonferrous metals industry. It is clear to me from the examples I have cited that in our haste to achieve our goals, we are risking the danger of undermining the fundamental institutions on which our society is based--our basic industry--and we are removing our ability to meet established goals in other areas! There is a strong likelihood, in my opinion, that the very steps which we are taking to preserve our quality of life could very well place in jeopardy the very quality we are attempting to preserve. Frankly, I see no winners if our present course of action is not changed. Obviously, we cannot afford to cure one ill if it means creating another...particularly if it carries with it the warning of former Secretary Simon of a loss in personal liberties. We simply cannot afford to see our industrial civilization collapse ...the consequences are far too great!

What we see happening is remarkably like the condition described by John Kenneth Galbraith in his book, "The New Industrial State". Mr. Galbraith perceives that modern industry, in the United States and elsewhere, is a collection of inflexible, self-satisfied colossi which must be brought "under control" by such strong medicine as regulation, anti-trust control, price control and, in the extreme, nationalization. I tend not to agree with Mr. Galbraith's views about modern industry and submit that his description is more that of industry in years gone by than that of "new" industry. His recommendations for the problem which he perceives are diametrically opposite that of what I would consider to be the correct approach. Mr. Galbraith's solution has been applied in many countries of the world with the result that these are now second rate countries. To this observer, the question "what can be done about what seems to be an inevitable downfall of all things we hold dear to our American way of life?" (That is, how can we reassure those unalienable rights spelled out in the Declaration of Independence ...Life, Liberty, and the pursuit of Happiness...in view of the many restraints being applied to the industrial base of our country?)

My solution to these problems may be naively simple for they are derived of a practical, not an idealistic, mind.

First, in my opinion, we must do everything in our power to return the image of American industry in the eyes of our people to the stature which it once held. A popular American concept (espoused particularly by our news media and our politicians) is that industry is nothing short of a "communist-like conspiracy" to undermine the country and to rip off the consumer. Industry is alleged to be the true cause of the "energy crisis", and the core of most of what is wrong with the country today. This is not "a collection of inflexible, self-satisfied colossi" we are speaking of, but the very backbone of our society--a vital component from which we gain our sustenance. Without it we cannot walk erect to meet the challenges of the future.

Second, we must modify our regulatory regime to accomodate to the cruel facts of reality. A physician is a poor practitioner indeed if in his attempt to cure the ills of his patient he causes the death of the patient. Regulations are a vital tool to establish expected and standard performance. However, if misused, they can be debilitating and even lethal. The basic industry of our country is too frail a patient to be tampered with in this manner. Standards must be demonstrably essential--not based on emotion and scant scientific evidence.

Third, whereever possible, regulations should be approached from the incentive point of view rather than from the punitive point of view. Competition and the freedom to innovately meet challenges is what made American industry great. There is no reason why these initiatives cannot also be used to fulfill society's expectations of industry. For example, why don't we apply the same medicine to the nonferrous metals industry which is being used on the automobile industry. Appeal to the competitive nature of the industry by taxing emissions by the pound just as "gas guzzlers" are to be taxed by their gasoline consumption.

Fourth, don't stifle innovation by pre-establishing not only the standards but the methods by which the standards are met. To do so is wasting America's most valuable resource--Yankee ingenuity, the ability to get tough jobs done with a minimum amount of red tape. The goals which have been established are tough ones to be sure but I, for one, would like to see how clever we can be in meeting these goals.

Fifth and last, we must restore the attitude of government to its rightful place of protecting the security and welfare of the people while helping to maintain an environment in which all of the instruments of society, industry included, can be free to fulfill their rightful function with a minimum amount of interference.

"And what of the future?" you may ask...I am hopeful, optimistic, and looking forward to the challenges it promises. I hope that you are, too.

CONTROL DEVICE SELECTION AND EVALUATION

(Panel Discussion and Open Forum)

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CONTROL DEVICE SELECTION AND EVALUATION

(Panel Discussion and Open Forum)

1. R. L. MEEK--Welcome to the panel discussion and open forum on Control Device Selection and Evaluation. For the past day and a half, we've heard quite a bit about control device operation and some of the difficulties encountered. Perhaps in some of the cases, they wished they had had a better method of selection and in some cases a little less severe method of evaluation. In any event, tonight is your opportunity to exchange views with what I consider to be a very distinguished panel. We have, sitting to my left, a panel with well over 100 years of experience in air pollution control technology. Our panel tonight is representative of industry, engineering concerns, and experts from the Environmental Protection Agency. Our ground rules for tonight are for you to direct your questions to the panel members, preferably by name, but if you wish, to the panel in general. I will not entertain any discourses or presentations from the floor. However, if you have any experiences that you would like to share with the panel and with the audience, these will be perfectly in order.

Our panelists this evening, seated from my left: James H. Abbott. Jim is Chief of the Particulate Technology Branch of EPA's Industrial Environmental Research Laboratory, Research Triangle Park, North Carolina. Dr. Seymour Calvert, President of Air Pollution Technology Incorporated, San Diego, California. Mr. Frank R. Culhane is Vice President, Sales and International Operations, for the Air Pollution Control Division of Wheelabrator-Frye Incorporated, Pittsburgh, Pennsylvania. Mr. Culhane is also President of IGCI, the Industrial Gas Cleaning Institute. Mr. E. L. Coe, Jr. is Manager of Advanced Technology for Western Precipitation Division, Joy Manufacturing Company, Los Angeles, California. Mr. Bruce Harris is a Sanitary Engineer in the Process Measurements Branch of EPA's Industrial Environmental Research Laboratory, Research Triangle Park, N.C. Mr. E. S. Godsey is Chief Fume Recovery Engineer in the Central Engineering Department of ASARCO, Inc., Salt Lake City, Utah. Mr. Sidney R. Orem is Technical Director of IGCI, the Industrial Gas Cleaning Institute, Inc., Alexandria, Virginia. Next to last, Dr. F. E. Templeton is Division Environmental Engineer for Kennecott Copper Corporation with the Utah Copper

Division at Salt Lake City, Utah. And, Mr. Lawrence C. Tropea, Jr., who is Director of Environmental Control for Reynolds Metals Company, Richmond, Virginia. Larry is also Chairman of the Environmental Committee of the Aluminum Association. So we have representatives from EPA and the three major nonferrous metals areas and for the major control technologies.

Jim, will you start the opening remarks.

2. J. H. ABBOTT--Dick asked us to tell you a little bit about what we do and what our background is to start things off. I'm Chief of the Particulate Technology Branch of the Industrial Environmental Research Laboratory located at Research Triangle Park, North Carolina. There are two of these Industrial Environmental Research Laboratories, one is located in Cincinnati, Ohio and one in Research Triangle Park, North Carolina. The one in Cincinnati is the laboratory that's sponsoring this conference. Our branch, the Particulate Technology Branch, is responsible for the development of particulate control technology not related to any specific industry. It's kind of generic work on various devices. We look at new technologies which we call novel concepts and novel devices, and we also do improvement and development work on conventional devices (fabric filters, electrostatic precipitators and scrubbers). Our R&D program is a rather large program; our budget runs something like 6 or 7 million dollars a year.

3. SEYMOUR CALVERT--Air Pollution Technology, or A.P.T. is an engineering company and our major competence is in fine particle control equipment design. Wet scrubbers are our principal specialty. We've done many EPA funded R&D programs, such as the wet scrubber system study which led to our writing the "Scrubber Handbook".

   We did several projects on flux force/condensation scrubbing (F/C scrubbing) which means scrubbing involving the condensation of water vapor. And we have studied the effect of that condensing water vapor to enhance scrubber efficiency. We have adapted existing theory, developed mathematical models, done laboratory work, two pilot plants, and two demonstration plants; one on a nonferrous metals recovery furnace and the other on an iron melting cupola. As a matter of fact, the second one is just about to start up; the construction is being completed on that now.

   We have done several projects on entrainment separators (or mist eliminators). We have looked at the theory and done pilot plant studies to determine the collection efficiency for the drops, the reentrainment limits on the separators, solids depositions, and methods of washing the mist eliminators. A.P.T. has done many field tests of large scrubbers on a variety of sources. The primary reason for doing these tests was because we, and other people, had developed methematical models for predicting scrubber efficiency, but we didn't have very much data to work with. In order to validate these mathematical relationships, we had to go out in the field and get data, and there were other people who have done the same thing.

A.P.T. has been working on high-temperature and high-pressure gas cleaning (by high-temperature and high-pressure, we mean up to 2100°F and pressures up to 250 psi). These studies are related generally to advanced energy products or processes and bear on the need to clean up the gas before it can be passed into a gas turbine. We worked on a basic study of particle collection under these extreme conditions and evaluated inertial impaction, diffusion, and electrostatic deposition. We have done a study on granular bed filters with and without electrostatic augmentation, and another on what we call the PxP, or particles by particles, dry scrubber; also with and without electrostatic augmentation. We're now working on fugitive dust control by means of electrically charged sprays in the spray charging and trapping (SCAT) system.

For commercial clients, we've done some projects on such things as sulfuric acid nucleation and condensation, especially in oil-fired power plants. We've done a study on power plant plume opacity as related to particle size and other particle properties. We have worked on flue gas desulfurization scrubber performance improvement and other projects of that type. Recently we have established APT Equipment Inc., a subsidiary, for the sale of air pollution control equipment.

We have made some proprietary developments on several designs, including what we call a Triple Scrubber which is a three-stage high efficiency scrubber which is especially well suited to situations where there is heavy dust loading and where the scaling tendency is severe. We've developed a spray scrubber which is especially good when moderate efficiency is adequate and when power requirement is extremely important. We have developed a high efficiency scrubber which in some ways surpasses Venturi scrubber performance. We have also developed some new mist eliminator designs for both vertical and horizontal gas flow orientation.

4. E. L. COE, JR.--I am with the Western Precipitator Division of Joy, a successor company to the Western Precipitation Company, one of the old line manufacturers of dust control equipment. I find it somewhat less comfortable than it used to be to speak to this industry inasmuch as the type of

equipment that we supply is now being forced down the throats of the industry. The previous arrangement where it was sold for an economic purpose is considerably more comfortable, at least in my view. I've been with Western Precipitation since 1963, all of that time in the R&D Department concerned with the development of mechanical collectors, scrubbers, fabric filters, and both wet and dry electrostatic precipitators. I would say that if there's been a change or is an ongoing change in the precipitator business, it's in the greater rationalization of selection and application of precipitators, principally as a result of the greater use of EDP equipment.

5. F.R. CULHANE--I am associated with Wheelabrator-Frye, a leading manufacturer of air pollution control equipment. We think of ourselves as one of the old guards in the business, having manufactured our first fabric filter for the foundry industry in 1913, and we became fully established in the nonferrous metals industry right after the second world war, specializing in the filtration of gases at elevated temperatures. Since 1913, after our beginning in fabric filtration, our product line has included shaking baghousees, reverse air baghouses, pulse fabric filters and Lurgi high-efficiency precipitators. In addition to our control work of particulate emissions, we have also done significant work in control of gaseous emissions, having pioneered the early work in dry scrubbing using limestone to scrub gaseous fluorides. This early work was performed by continuously seeding a bag filter with limestone on a ferrous open hearth facility in the early fifties. This was followed with a dry scrubbing pilot unit at a western aluminum smelter. This work was the fore-runner of dry scrubbing florides from aluminum reduction plants. We are licensees of the Alcoa A398 process. In addition, we pioneered the first gaseous control of SO_2 using spray drying as the reactor in the power industry. We have also done activated charcoal solvent recovery work. In addition, we have done control work on what could be called gunk, not gaseous or solid particulate emissions, but liquid aerosols which is somewhat of a no-man's land and something that is not talked about too much. I should also add, that in addition to manufacturing this equipment, we also manufacture filter media. We have three plants in this country, one in France, one in England and a couple in Canada. Internationally, we manufacture equipment either through equity holdings or license agreements in Canada, Mexico, Brazil, England, France, South America, Australia, India and Japan. In order to serve the nonferrous industry, we have become involved as engineers in the entire process.

6. E. S. GODSEY--I started working for ASARCO, known at that time as American Smelting and Refining Company, in 1949 at the smelter we have in El Paso, Texas, as a Metallurgist.

The El Paso smelter is a primary lead and copper smelter and also recovers zinc from blast furnace slag in a fuming furnace. Part of my duties as a metallurgist included the day to day supervision of the baghouses and Cottrells in the plant. Eventually, I was transferred to the Salt Lake engineering office and to the Fume and Dust Recovery Section, and in 1973 became head of the department within the Central Engineering Department. Part of my duties include selecting the type of device which will be used in a particulate recovery system, whether it be a baghouse, Cottrell, scrubber, or whatever, writing specifications for the equipment we buy from vendors, evaluating the bids, selecting the equipment and ultimately running stack type tests to determine if the efficiency meets the guarantee or if it meets the regulations.

7. D. B. HARRIS--I've been with EPA for ten years. The last eight I've been involved in particle measurement development in support of the other branches in the Industrial Environmental Research Laboratory, as well as the other sections of EPA throughout the country. Special interest has been particle size measurements as we relate to control device evaluation, in direct support of some of Jim Abbott's work in developing performance models and testing novel devices. We have evaluated the performance of the measurement devices themselves, including impactors, cyclones, and some automatic instrumentation. For the last two years, I've been involved as project officer for the methods developments for fugitive measurements. We're working on a Fugitive Assessment Sampling Train (FAST) for these that involves cyclone and probably elutriator sizing with backup filter, as well as a small organic module to collect vapors from fugitive sources.

8. S. R. OREM--The Industrial Gas Cleaning Institute is the national association of manufacturers of air pollution control equipment. It was founded in 1960 with 16 charter members, and it now has 42 members. As our chairman mentioned earlier, Frank Culhane is in the middle of his term as President of the association, and has previously served as the Fiscal Officer and Vice President. In the last year, last summer in fact, we relocated our headquarters from Stamford, Connecticut, to Alexandria, Virginia, for what I think must be obvious reasons. The association is divided in several ways, perhaps one that is of more interest to this group is by product categories. We have six divisions: an electrostatic precipitator division, a fabric filter division, a wet scrubber division, a flue gas desulfurization division, a mechanical collector division, and finally, a gaseous emission control division which embraces such products as flame afterburners and catalytic oxidation equipment. These divisions meet twice a year, usually having an outside speaker on some timely topic in the morning and a business session working on publications, test procedures,

or something of that sort in the afternoon. We also organize and sponsor seminars on specific topics. Two recent topics were a seminar on explosion protection in air pollution control devices. This brought a turnout of about 200. An earlier seminar on construction problems, erection problems, again brought a very promising turnout. The big affair of our year is our annual meeting beginning the end of next week in Phoenix. We have the usual standing committees to deal with such things as preparing a budget and policing it, government relations, industry relations, publications, etc.

We are proud of some work we have done for the EPA, totaling, I believe, 6 contracts since 1971 which essentially were study contracts to (1) identify a preferred control technology for a particular problem, and (2) develop costs, installed costs, for that preferred control equipment. One of the important functions that we serve our members, and for that matter, the public, is to measure the market. We compile quarterly reports of new orders booked in the market.

Of particular interest to our topic tonight are some of the publications that have been prepared by task forces over a period of some years. For example, a bulletin on information for the preparation of bidding specifications on precipitators. A bulletin on a bid evaluation form for precipitators, bulletins on types of equipment and terminology commonly used for fabric filters, for wet scrubbers, and for mechanical collectors. Those are some instances and although some of you, I'm sure, have used some of these bulletins, I believe they have not been as widely exposed in the nonferrous metals industries as they have in the electric power, fly ash, side of the marketplace.

9. F. E. TEMPLETON--I manage a department in Kennecott's Utah Copper Division that's responsible for all aspects of the environmental control program of the division, including air, water, solid wastes. We do such things as identifying problem areas, developing solutions, and monitoring emissions for air, water, and solid waste to make sure the division is in compliance with regulations. Kennecott just recently completed a $280,000,000 program at its smelter to control air emissions. This includes both particulate and sulfur dioxide. We are proud of the program and feel it is a successful one. We are meeting the standards, both the particulate standards and the SO_2 standards and I guess I'm a little chagrined to read in this EPA bulletin, "Nonferrous Technical Awareness", that we're still having problems. I'm not sure if the problems are all technical ones; I think there are some regulatory ones, but we think the job we have done in selecting the course of action to take and the outcome of it is the correct one, and we hope as time goes by, we'll be able to persuade some of the regulatory elements

of EPA and the state that we're correct.

10. L. C. TROPEA, JR.--I manage the environmental control program at Reynolds which encompasses the air, water, and solid waste areas. In addition, I have responsibilities for the corporate part of the fire protection program and the noise control engineering program. Activity-wise, I am chairman of the Environmental Committee of the Aluminum Association, and also participate in the United Nations Environmental Program consultive group on the aluminum industry.

    I'm not heavily involved in the selection of equipment, but I do get some involvement in that area. Most of my time is spent in the environmental policy area. The major part of my activities involves interfacing with EPA. Also, recently I have been involved with Clean Air Act Amendments of 1977. That really pretty well summarizes my background.

11. R. L. MEEK--Okay, we'll throw the floor open to questions, comments, observations. Please share your experiences with us. I ask that you not speak from your seat but use one of the two mikes in the center aisle. Please go to one of those two microphones. Identify yourself and your company and ask any questions or comments.

12. ROGER W. LEE--Imperial Smelting Processes-I'd like to direct two questions to Mr. Culhane as a supplier of equipment, and subsequently, to Mr. Godsey as a user of equipment. Both of these questions relate to bag filters. I'd like, if possible, an assessment of the relative roles, bear in mind today's technology, of the shaker type, reverse air, and pulse jet filters. That's the first question. And the second one relates to cooling of gases before bag filters. Have you done any recent work to evaluate, respectively, zig-zag coolers, forced-air coolers or water-cooled ducts for cooling gases before putting them into bag filters. Also, have you done any work on use of high temperature filters, for example, stainless steel or ceramic-fiber filter bags.

13. F. R. CULHANE--That was a planted question, because last night that question was asked of one of the speakers and after the session was over, I suggested to Mr. Lee that perhaps he should ask the panel that question. The question of why not a shaker-type or why not a pulse-type is a question that I've heard many times. We have product people within Wheelabrator-Frye that are working on different product lines, be it the pulse jet fabric filter, the shaker baghouse, the reverse air baghouse or the electrostatic precipitator. The question of which collector is the best for a given emission problem comes up frequently, and we have found that the best answer to the question is economics. We all have personal preferences; and with our preferences, we have prejudices,

particularly when we are dealing with devices that we are comfortable with, devices that we know well, and ones in which we have a high degree of comfort, and our fears direct us into dealing with known successes. The first step to the answer is the proper and accurate identification of the emission problem. You could almost make a lifetime study when you get into the critical areas that affect the size of equipment, such as particle size distribution, particle chemistry, gas volume, turn down ratio, temperature, gas chemistry, and resistivity of the dust. As soon as the problem is well identified, then the equipment can be properly sized and the capital costs established. In addition to that, capital cost is necessary to run an analysis on the operating and maintenance costs, utilizing accurate cost values and recognizing this variation of these costs. For instance, the cost of energy is different in the northwest than what it is in the northeast. This sort of cost analysis will give you direction in the selection of the best kind of equipment for a given problem. We've recently done this on a major project just a month ago, pulse versus shaker. The pulse collector first costs were much more attractive, but the operating costs in time ate up the capital savings. The particular funding of the project was such that it was attractive not to realize the initial savings in first costs and to minimize the operating and maintenance costs. The shaker baghouse, in this case, was accordingly selected. I would like to make another key comment on this question. You have a countryman, Mr. Lee, by the name of George Bernard Shaw, who said, "Generalizations are not worth a damn, including this one." That is the crux of the problem and Mr. Englebrecht said it well yesterday, that he would have a hard time giving a specific answer to a general question. But our best answer to your question, Mr. Lee, is that we would make that determination on economics. On the other phase of the question regarding gas cooling, perhaps Mr. Godsey would like to respond.

14. E. S. GODSEY--You mentioned the different types of pulse shaker or reverse air cleaning. ASARCO has traditionally used the shaker type baghouse. The oldest one that I know of in the company started in 1902 and is still operating. We have traditionally used the shaker type and in this application we hold the air-to-cloth ratio as low as 0.5 to 1. This is a large baghouse built for an expanded operation which is now reduced in size but with the same bag surface. Our normal design ratio would be about 1.5 to 1.7 to 1 on the shaker type. We are now getting more into the pulse type cleaning, and we started using them primarily because of space restriction. We couldn't get enough bag surface within the alloted area with the shaker type so we went to the pulse type cleaning. I, personally, am completely happy with all the pulse type applications that we have. Some are on sub-micron

particulate. In these cases we are extremely careful to specify not more than 4-to-1 air-to-cloth for the pulse type. On cooling, traditionally we have started out at 300 and gone to 500 psi water. We are now slowly converting to air atomizing primarily because we can use some of the blow-down water from other processes for cooling. We cannot use pressure sprays with dirty water, but we have had reasonably good success using air atomizing sprays. The grain loading on our baghouses compared with what I hear in other papers is quite low, generally about 2 grains per standard cubic foot. Although in some of our zinc-fuming furnaces, they can go as high as 15 grains per standard cubic foot. The higher grain loading is on a shaker type baghouse and we always build at least one extra compartment, as someone mentioned this morning, for the cleaning cycle and again for maintenance so that we can take off the stream and still have adequate capacity to ventilate the furnace that it's connected to.

15. F. R. CULHANE--I would like to briefly add the question of gas cooling. Typically, we use several means of cooling the gas for fabric filter applications. One of the problems in selecting a system for a given problem is thermal control. We have used air dilution, bleeding in outside air to cool the gases. We have used spray towers utilizing high pressure water to atomize the water in order to completely flash evaporate the water to water vapor resulting in operating evaporative cooling towers without a run-off. We have used waste-heat boilers, air-to-air heat exchangers, radiant coolers, such as the trombone coolers which are so characteristic of the nonferrous industry.

    With regard to your question of our work on high temperature filters, commercially, the highest operating fabric filters today employ fiberglass at 550°F. We have looked in the laboratory at various materials that would withstand temperatures in excess of that, such as sintered and woven stainless material as well as ceramics. One of the major disadvantages when we get into operating at high temperature is that under expanded conditions we have more gas to filter, so we're losing ground when we get into temperatures above 500-550°F. In addition, we get into special housing materials and hardware because of higher temperatures of the gas.

16. CARL H. BILLINGS--Arizona Bureau of Air Quality Control-For the past two years Arizona has had no particulate standards. We've stayed our particulate standards for copper smelters. We've been carrying out a program to try and determine what the standards should be. The data are just beginning to come in during the last three or four months and we've been attempting to analyze our data and it's a little difficult. We have removals from 74.5 up to 99.9%. About half of the installations do not meet the standards we previously had

which were the two EPA curves sometimes known as the San Francisco curve and the Colorado curve which are process weight curves. My question tonight is: Given this wide variation of measurements, how is one supposed to evaluate what is proper control technology? I don't care whether it's best available technology or reasonably available technology, but what is proper control technology for electrostatic precipitators on copper smelters? And I direct it to the whole board. I'd like to hear a discussion of it.

17. E. L. COE--I think that's a question that has as many answers as there are people in this room. The performance of equipment on a copper smelter or on any other application depends on not only the initial sizing of the equipment but on the way that it's operated and maintained, and on the way it's tested. I think that you've cited some very low figures which may either relate to very old equipment or may very possibly relate to equipment which was designed to be tested by the ASME method. Very possibly, different results were obtained when tested by other methods such as the EPA Method 5.

I think as far as electrostatic precipitators are concerned, they've been applied in the smelting industry for years, and their performance and faults are well known. I think that I am beating around the bush as far as your question is concerned and I think that I have to do so because there is not a single definite answer. As Mr. Englebrecht pointed out fairly well, you can design a precipitator as large as you need and it'll catch as much as you wish. Bag filters have well-known high performance capabilities for collection of particulate, but they also have limitations. So does any other piece of equipment that you can think of. If you can guide me through this maze that I am constructing here, as to what kind of path you want followed toward your answer, perhaps I can be of greater assistance.

18. F. R. CULHANE--You know it's really an interesting question, because you can carry things to an extreme from the standpoint of efficiency, 8 fields of precipitator, high SCA's, 800, 900, 1000 square feet of collecting surface per 1000 cfm. As engineers, if somebody gives some requirements as to how clean we must clean the gas...we can provide the means to achieve the required results. In the case of beryllium, we have installed baghouses in series. The first baghouse removes 99.9% and the second one we pre-coat with a filter aid and get another 99% out of the 0.1% that we lose in the first unit. Is that BACT? That would scare me. I think we really should get into BAPCT--Best Available Practical Control Technology. Perhaps that's a question best directed to the EPA.

19. R. L. MEEK--Bruce, do you have any comments on that broad range of test results?

20. D. B. HARRIS--I think when it comes to evaluations of specific control devices, Jim Abbott would be able to comment more on that. If he has questions on test methods, I'd be glad to comment.

21. J. H. ABBOTT--Well, I'll say that you haven't really told us very much except that you've been out testing equipment and have a broad range of results. I'm not sure what you were looking for when you tested it, but certainly you can go in the field and find as broad a range of efficiencies as one wishes to find depending on how poor or how good the equipment that you test happens to be.

22. C. H. BILLINGS--To begin with, you are probably all familiar with the development of process-weight type curves and the way that the so-called San Francisco curve was developed whereby they did go out and test a number of industries, and then they created an envelope curve that included all those industries. All the industries tested therefore passed. We attempted to do the same thing with copper smelters and of course we haven't succeeded. We even developed our own test method to obtain consistent results, and in fact, it did give more reproducible results than Method 5. When we began this study we found out that Method 5 was very inconsistent due to the generation of acid mist in some of the ducts and one thing and another in these copper smelters. Therefore, we've developed a test method in which the hot box was kept at a higher temperature and all the condensables were supposedly carried through the filter and collected in an impinger. So that all we were left with was mineral particulate. So, we feel that our methods of testing, and Mr. Godsey was on the panel that supervised some of these tests, were well done and that they were done with about as good technology as you probably could muster. The only other remark I could make is that I am virtually certain that 74.5 is too small an efficiency for an electrostatic precipitator for only one reason, that if I've got to go that low with an electrostatic precipitator, I'd rather replace it with a multiple cyclone and save the electricity.

23. E. S. GODSEY--There is no such thing as a typical smelter. But within our operations, we cool the gas to about 250-275°F, particularly if we are going to stack. If we are going to an acid plant, we have some that operate in the range of 600°F. As you say, the volatiles go through the Cottrell. They have no chance of being recovered unless they are as a solid particulate. Now, if the company that has volatiles going through is forced to cool down to improve the efficiency, they are going to condense the acid mist; it's going to be

recovered in the Cottrell, and you can almost stand there and watch it melt in front of your eyes. I've done that. I've watched it. Not day by day, but go back in two or three months and see holes in it that weren't there before. So the acid content is very important so far as Cottrell efficiency is concerned, and too much, you're spending an awful lot of time with the unit down to consider the gas stream in addition to the efficiency of the Cottrell. As I say, there is no such thing as a typical one. Each one has enough difference in it that it's a special problem in itself. In ours, we have some that we couldn't lower the temperature to 250 or 300°F. It would kill us. Others we have do operate in the low temperature range, and generally speaking, on these we get better efficiencies than we do with the ones operating at the 600° range. We started out with a Cottrell purchased from another company, not one of our own design, and operating it at the temperature recommended which is about 400°C. With a copper converter operation, you can't maintain 400°C. You can't maintain any specified temperature. The intent was to operate it at 400°C, reduce the resisitivity of the dust, and get what they indicated as 95% efficiency. It turned out to be more like 85%, primarily because we couldn't maintain the temperature. So, in our case, we changed the flue configuration, we added water cooling, and we improved it up into the 90%. But, again, this is one going to an acid plant, and efficiency is not as critical as if we were going to stack with it.

24. F. E. TEMPLETON--I'd just like to support Ed's point. It's wrong to generalize all copper smelters, and I think the notion of the so-called curves and the process weight limitations applied across an industry is really the wrong approach to take. Each situation is unique, and the results that we've gotten indicate that you get a pretty broad range; with any given control system you're going to get a broad range of emissions. You'll get a cluster of values around an average sort of area, but you'll get some higher values and some lower values. If you do enough tests, you'll end up with a distribution curve. So, if you ask what the efficiency of the control system is, it really depends on when you measure it. There will be a maximum and a minimum and, as I mentioned, an average. To try to say that all plants should be able to achieve x percent control using a certain kind of precipitator unit just won't work. It just isn't appropriate to put a fixed emission limitation on all smelters of the same size or proportionate size and try to develop a regulation that way. I think what you've got to do is to have a regulation that allows for the variations in the process and in the units that control the process gases.

25. J. H. ABBOTT--I'm still not sure that I totally understand the question, but I do have a comment. Regardless of how

good your test method may be, when you go to a precipitator at a given plant in a given situation to make test measurements, before you factor the data into any kind of standard or regulatory plan or whatever it is you're going to use it for, I think you have to apply some good engineering judgment to the data. First of all, you have to examine the precipitator, what it was designed to do, what it is doing, what are the inlet conditions, what are the outlet conditions, what are the characteristics of that dust, the resistivity, the particle size distribution. There are mathematical modeling techniques that one can look at to try to explain an anomaly that you might have seen, and I think in some cases you might want to reject some of these points as not being typical. You might want to utilize what one would consider to be the typical points in whatever stragety you might be utilizing or attempting to develop with this data. I don't think you can just go out and blindly take a series of points and then factor them all into something, whatever it is, and come out with anything useful. There just has to be something more to it than that.

26. F. E. TEMPLETON--We've had EPA come out and make measurements on our plant and there have been times when they've turned off the measuring equipment because they didn't feel enough units were running, or it wasn't typical, and I guess my reaction is that you'll never have a typical condition. It's what it is and I think it's a little bit misleading sometimes to try to achieve a single number that expresses the entire control scheme. I think Arizona right now is doing some work with its multi-point emissions limitations concept. I don't see any reason why that same concept, for example, couldn't be applied to particulate emissions because you're going to get the same kind of results whether the technology is for SO_2 control or particulate control. You're going to have a distribution around an average and if you specify the number as the average, no matter when you go out and measure, you'll probably never measure it right at that particular time. You'll either be above it or below it. So, rejecting the measurements you make, when you go out and do tests in order to try to find some typical condition, I think is just a fruitless search. I don't think you'll ever find it.

27. E. S. GODSEY--I think we've found that in our operations, too. Absolutely, anything that happens is typical. The best way to get the reverb arch to fall in is to put a pitot tube in the flue system. I know, I've done it. And, breakdowns are common. They're part of the job, part of the operation. Maintaining maximum smelting rate for the four-hour period that it takes to collect a sample, at least in our flue system, it's extremely difficult to do. You have not only mechanical breakdown, but you fill up the hoppers, you fill up the reverb, the converters are down. All of this is

typical operation but it's not maximum smelting rate. Makes it very difficult to sample at maximum smelting rate over a four- or six-hour period. I agree, I have to take the ups and downs and try to estimate or guess what it would be at full smelting rate and that is extremely difficult. We don't smelt at full smelting rate long enough to call it a typical operation.

28. J. H. ABBOTT--I would like to say that I think you'll have to agree that a precipitator or any other control device, and there are those operating this way, that perhaps would have half the amount of plate area that it really needed for a given situation. Maybe one that was designed in early 1900 and still operating today certainly couldn't be considered typical. I don't think we at EPA could accept that as being typical. There are good units operating well, and there are those that aren't and that's what I had reference to in my comment.

29. JOHN O. BURCKLE--EPA-We've run some particulate emission control tests on some southwest copper smelter ESP's, and we found a great deal of difficulty getting really good sampling conditions at the inlet of the precipitator. I would suggest that some of your problem would lie in that area, getting a good inlet sample, both in flow measurement and the problem of getting a good isokinetic traverse across those cross-sectional areas from which you are trying to obtain your samples.

30. KONRAD SEMRAU--SRI International-It occurs to me that at least part of the question here revolves around definitions of particulate matter. In some instances, particulate matter is assumed to include materials condensable down to a certain point. If you have a device that's operating at elevated temperature, there's no point in calculating its efficiency on a basis that would require a temperature much lower than any part of the system is operating. Because, quite plainly, this device, whether it's a bag filter, precipitator or anything else, is not going to collect material that is not particulate at the temperature of operation. If you have a condensable present, and you want to specify a control efficiency with reference to that component, it's going to have to be defined in a way that can be interpreted in terms of the performance of the unit or the temperature of operation of the unit.

31. C. H. BILLINGS--In the beginning of my remarks, I think I stated, but maybe not, that we had developed a method that, in a sense, rules out condensables. Our method of testing, theoretically, collects all the condensables in impingers. Then if you wish to add them back in as a regulatory measure, why that's fine, but you're really only collecting on the

filter what we call mineral particulates. So this would rule out these data that were not taken on condensables. What we do with it later, as far as regulation, is something else. The other item was on the basis of the difficulty of getting inlet sampling. We did have difficulty in getting inlet sampling, but we have cases, for instance, where one of the precipitators that we're dealing with was designed to meet the more stringent of the process weight curves, and a great deal of money was spent to change that precipitator from meeting the old process weight curve to the new process weight curve when it came along. And this precipitator on the outlet has never even come anywhere near meeting that curve. So not all of our problems were on the basis of inlet and outlet.

32. CHARLES D. HENDERSON--St. Joe Lead-I would address the question to Mr. Abbott and Mr. Harris, but I'm not sure that they're the appropriate ones. In our industry, as you know, we've been forced by recent regulations to look intently at the exit of collection devices. We believe in order to capture fugitive emissions, whether you use an electrostatic precipitator, a bag collector or a scrubber, you're likely to get exit emissions in the neighborhood of 0.01 grains per cubic foot. We calculate this to be about 23,000 micrograms per cubic meter. Then, if the dust is typical, it contains 50% lead. My question is this: how far away, in terms of horizontal or vertical distance or a combination of both, would the exit of such a collector have to be from your measuring point in order to meet the EPA standard of 1.5 micrograms?

33. D. B. HARRIS--Well, I'm not a dispersion modeler, and I think you're probably going to have to go to them to get your answer for that. Sounds like a long ways.

34. J. H. ABBOTT--Also sounds like an awfully low collection efficiency for a baghouse or for even a good scrubber. The figure of 0.01 is a fairly high emission.

35. C. D. HENDERSON--In the survey that Mr. Caplan gave us during yesterday's session, 0.01 was about in the middle. Actually there were some that were 0.05 and higher on the exit of the collectors in his industry survey. Some of them were listed at 0.001 and I would personally question that quite seriously as being a practical, good operating collector.

36. D. B. HARRIS--I don't see why you question that. We've seen some efficiencies that low. I can't say specifically for lead operations—it's been five years since I've tested anything like that. But for other operations, some times when we're testing we can calibrate our equipment better using the exit exhaust of the collector than we can the ambient air.

The collectors are working that well.

37. J. H. ABBOTT--They're operating at .00X on fairly fine fumes. Now again, I don't know that much about the lead problem that you speak of. It seems that's an awfully high emissions rate.

38. C. D. HENDERSON--Maybe Mr. Culhane and Mr. Coe can give us some guidance on that. What kind of exit dust conditions would you be willing to certify for your customers?

39. F. R. CULHANE--Well, there is a difference in fabric filter performance in the laboratory and in the field... and a difference between a fabric filter that is tuned up and one that is not. There are three major size baghouses at Herculanium and a fourth smaller one with a concrete casing. As the bags fail, they have to be replaced, and it's possible you may have some bags that need replacing and under those kinds of conditions the emission values could get up perhaps to the 0.01 value. I think the maintenance at Herculanium is outstanding, and I think you've got an outstanding crew. Theoretically, we should see two zeros day in and day out. I would say your outlet should be in the 0.007-0.008 range.

40. C. D. HENDERSON--Well, all right. The difference between 0.007, 0.008, and 0.01 isn't all that great. I was just seeking a number to be typical of a well-operating collection device that's been out in the field for three or four years. And, further, of course there are many applications where a fabric filter is not appropriate because you are handling a wet material and you have to go to scrubbers, and I believe that the 0.01 is not a bad description of a scrubber and might even be low for the typical scrubber. The question still remains, whatever the collection device, you're talking at the exit of the collection device, dust loadings in the thousands, and you have to go to 1.5 micrograms at some distance away where the measurement is going to be at the fence line.

41. F. R. CULHANE--This is outside my field, but it would seem to me that the height of the stack and the distance from the plant are the critical values that would establish the relationship between stack outlet emissions and ambient background emissions, along with plume buoyancy, wind velocity and topographical conditions.

42. C. D. HENDERSON--I realize that the true solution is a modeling problem but I thought amongst the experts on the panel there would be some locus of distances and elevations in typical environmental and meterological situations.

43. R. L. MEEK--I don't think we have any modeling experts on the panel, Mr. Henderson, unless I'm mistaken. Do any of you

gentlemen have that kind of expertise?

44. F. E. TEMPLETON--Well, I'm not an expert, but I do have some insights. We've done a lot of modeling studies, and been the recipients of a lot more. I don't think there's any fixed answer to the question. It depends on the circumstances, on the terrain, the frequency of occurrence of inversion conditions, and a lot of meterological factors that are totally different from one area to another. I can tell you this: depending on the terrain you're in, a 1200-ft stack will sure do a lot of good. I don't think there's any given answer. If you have really high emissions, and they are near the ground and at locations relatively close, within a mile of the plant, you're going to have high concentrations.

45. D. B. HARRIS--My suspicion would be that your low level emissions are the ones that you're going to have to worry about to meet your fence line fugitive emissions level, not your stack emissions. You're going to have to worry too about your slag dumps and storage piles and such things to meet 1.5. The stack is not going to be the one that's going to impact that standard.

46. F. E. TEMPLETON--If your fence line is close to the plant, fugitives are going to be the major problem you are going to have.

47. C. D. HENDERSON--Just in response there, to control the fugitives, where you have particularly moist dust situations, a typical thing is a scrubber with normally an outlet stack on the order of 15 meters or something of that sort which is fairly low level compared to a 1200-ft stack. And with the scrubber discharges, you have the additional problem of the power, reheating, or whatever is necessary to get it to move up the stack and to keep it from tumbling right down to earth again as soon as it gets out of the stack.

48. R. L. MEEK--Anyone in the audience have any experience in dispersion modeling that might contribute to this? I don't think our panel members are dispersion modeling people.

49. J. H. ABBOTT--Dick, I'd like to ask the gentleman a question, if I might. What is the inlet loading to the devices that you're speaking of here? What is the concentration at the inlet?

50. C. D. HENDERSON--On the order of 8 to 12 grains per cubic foot.

51. S. CALVERT--I don't really know what your specific question is about dispersion, but, in general, it isn't just a matter of a dilution factor, but you have to know the mass rate of

emission. If you know the kilograms per hour being emitted, the stack height, the wind velocity, the turbulence factors for the atmosphere, and any other special conditions, you can predict ground level concentrations at different points. I don't have a chart here to do it or I'd do it before your very eyes, but I'm not sure I understand what the question is.

52. C. D. HENDERSON--I guess my question is really to point out what I feel is a dilemma in our industry in that we have a typical sinter plant, and in an area of 20 meters square you may have 3 or 4 or 5 or 6 fugitive transfer points and locations and collecting these into scrubbers or some other device to collect the dust, each of which has an exit at the loadings we've been talking about. And with this concentration of necessary devices in order to keep the internal part of the plant clean, we are moving to the outside ambient air. And then this poses a problem, even with well run operating clean-up devices, we are emitting into the ambient air fairly high concentrations of particulates and lead. This then puts a quite practical limit on how far away from a fence line these devices can be located. Quite often, depending upon the geography of the plant location and the territory around it, you soon go through your modeling and your examination of the numbers, and you find that you don't have enough property in order to possibly meet this. We're into technical problems that appear to me to be quite severe for our industry.

53. L. C. TROPEA--I'd tell you that maybe your answer lies in litigation rather than in dispersion. Second, there has been a great deal of discussion about ambient standards being applicable at plant fence lines. I might point out that there have been some documents in circulation that have been pretty quiet for a while which propose to change the traditional definition of ambient air that would have rather interesting ramifications if you have to meet that standard inside your fence rather than just at your fence line. So, land acquisition may not be the answer.

54. R. W. LEE--I would like to support what Mr. Henderson has said. I think his figures are very typical, in fact. He's talked in terms of 101 grains per cubic foot, which is 23 milligrams per cubic meter and as he says, 50% for lead content is quite typical. This ties up with, for example, the British Standards for lead which are 12 mg/m^3, Japanese standards are either 10 or 5. So the figures he's talking about are really quite typical of standards elsewhere and what is achievable on lead practice. Normally, I find, which I think he is referring to also, that as far as I know, the EPA standards for primary smelters is 50 mg. So we are talking about much lower levels than the EPA levels, anyway.

Then you try to relate this to this fence-line standard which is 1.5 micrograms per cubic meter and you've got quite a problem. Our experience is that, in fact, the maximum impact at ground level is about 10 times the stack-height distance away from the stack. So, if you're talking about a 60 meter stack, under our sort of conditions, anyway, which may not be typical, you're talking about maximum impact on the ground about 660 meters away from the stack which may or may not be within your fence line. What determines what happens to your fence line is more likely to be these fugitive emissions which are windblown dusts of one kind or another which represent a serious problem if you are ever going to get down to the levels which are being talked about. But, like many people, I would dispute the basis on which the EPA has selected this level of 1.5 micrograms per cubic meter. It seems to bear little reference to actuality. There seem to be a number of imponderables added together, the end of which, 1.5 is the allocation given to the smelter. But I think it includes such things as the allowance in the air from other sources and so on. It's really quite a complicated addition of one thing and another. Also, it doesn't seem to take into account either the particulate sizing or fugitive emissions at the fence line or the chemical composition. You know, generally, in the industry it has been regarded that sulfide lead is not particularly harmful; it doesn't get ingested, but as far as I know, the EPA standard doesn't take this into account.

55. FRED (A.B.) CRAIG--EPA-In reference to Mr. Henderson's (of St. Joe) question, he referred to fugitives being a problem versus the process emissions. Mr. Calvert, I believe, when he gave his opening remarks, made some mention to the use of charged fogs, or something of that nature for dust control. The discussion we had following the paper I gave on Monday morning centered around dust disposal. What I'd like is for Mr. Calvert to make some comments as to the use of charged sprays with regard to the type of problem Mr. Henderson mentioned in order to find out if there is a possibility that maybe there may be an option or an alternative.

56. S. CALVERT--The project on charged sprays on fugitive dust emissions is a new one, and the equipment is just about built now. So I can't really comment on our work. There has been work done at the University of Arizona with charged sprays directed at the fugitive dust source, and they have had reasonably good results with the control of several types of fugitive emissions.

57. J. M. HENDERSON--ASARCO-I don't have a question but I do think it would be appropriate in connection with your question, sir, to remember that the new source performance standard for lead smelters, blast furnaces, tip end of the

sinter machine, etc., is 0.02 grain per standard cubic foot.

58. K. SEMRAU--I would like to address this question to Mr. Culhane, but with comments by anyone else who wants to offer them. What has been your experience in the pulse jet type filters with the use of either woven or felted type fabrics? A good deal has been made in the past of the importance of the dust layer as a filter medium in itself and with the woven type fabrics, of course, the reverse jet tends to disrupt that. Can you give us any information or leads on your experience with the two types of fabric?

59. F. R. CULHANE--Well, the principal with the woven fabric is that the dust does the filtering. The interstices between the threads or the fibers of the cloth will allow initial seepage until the cloth heals and you establish a dust cake. As you know, we collect particulate below submicron in size with a fabric filter. The finest particles that I have witnessed collected with a fabric filter are particles that have been measured in Angstroms, 10,000 Angstroms to a micron, and we've collected particles as fine as 320 Angstroms. The only reason that we can collect this fine particulate matter is because these particles, fortunately, agglomerate, and in agglomerating they form an excellent filter cake achieving high filtration efficiencies in the 99.9% range. When we get into a pulse-type fabric filter, our design goal is to increase throughput at comparable filtration efficiencies, and a tight dust cake is a limiting factor in achieving our goal. Accordingly, felt is substituted for the combined woven fabric and the dust cake. The felt requires more energy to clean and the deacceleration forces effectively clean the felt on a frequent sequence. Substitution of woven fabric in a pulse jet will result in over-cleaning and loss of the filter cake and reduced efficiencies.

60. MARK HOOPER--EPA-One of the things I would like to comment on is that I think the lead standard was promulgated as a result of a court order handed down to EPA within the framework of Section 307 of the Clean Air Act. The lead industry does have a forum to bring suit, and I think they may have done that in the Circuit Court of appeals for the District of Washington, D.C. I would imagine and hope that the lead industry can develop a good case at least to bring forth their concern. I'm not qualified to say whether that standard is right or wrong. But the forum is there for the lead industry to speak its piece and hopefully, at least, if they have a valid case they could convince the court. I believe that's where it is now.

61. GEORGE THOMPSON--EPA-I think in the wastewater area, we're coming up with new techniques that are very good in advancing the state-of-the-art. We have some good ideas in the solid

waste area. Considering the expertise on the panel---the panel being represented very well by industry, manufacturers, and the EPA itself---I ask, where do we go in the future? What's forthcoming to advance air pollution control technology?

62. J. H. ABBOTT--That's the topic of the paper I'm scheduled to give the last thing tomorrow. So I'll defer my answer until then.

63. F. R. CULHANE--That's a question that I hear often. We are still using the same basic principles of fabric filtration that Mr. Godsey tells me were first employed in 1850 in a zinc operation along with the work in the early 1900s. Also, the early work Dr. Cottrell did in the first decade of the century employed the same electrostatic principles that we employ today. It's interesting that a lot of our gas cleaning experience today evolved out of the nonferrous mining and metallurgical industry. With 6-8% of your material going up the stack, your industry has some economic motivation to do something about it. In those early days of stack gas cleaning, basic laws of filtration and precipitation were applied to gas cleaning and they're still in force today. We have not had a major breakthrough in basic principles since the first Cottrell in 1912 or the first baghouse in 1900. We bump into the same drag loss in a fabric filter for a given filtering velocity. We are fighting the same migration losses that were fought in the early days. The progress that I see in the gas cleaning industry is the application of the control equipment. In this area, our progress has been outstanding. Certainly, there has been work done in the refinement of the equipment and progress has been achieved in materials, better electrical controls for precipitators, and after the Second World War, the synthetic fibers opened up new applications of fabric filters for us as we were able to operate above the dew point at 300°, 400° and 550°F. But our principal progress has been in the application of this equipment.

What's around the corner? Something that can cut the size of the precipitator in half? Or cut a fabric filter in half? Or cut the energy requirements of a scrubber in half? I don't see that sort of a thing in the picture if we're looking for that as progress. You're in R&D, Mr. Coe. What are your thoughts?

64. E. L. COE--I concur, at least in part, with what Frank Culhane says in that most of the possibilities for improving particulate collection appear to lie in the enhancement of the basic physical principals that we've been working with. There is work going on to decrease the size of the precipitators through the use of a better charging mechanism for the

precipitator. This work is sponsored by the EPRI and is taking place at the Arapaho Station of the Public Service Company of Colorado. It has promise of reducing the size of precipitator required for a given performance. There is a good deal of work yet to do on that project before it's determined whether it is in fact a God-send or an also-ran. Similarly, enhancement of fabric filter collections through electrostatics in combination with scrubbing has been investigated, and I believe that Dr. Calvert is one of the prime investigators of such phenomena. Unless there is a genius who has invented a new physical principal that we've not discovered, we're going to collect dust in the same way, but I also believe that we're going to improve its reliability.

65. S. R. OREM--Having watched this business since 1937, I like to feel some remarkable progress has been made. I think particularly of the number of power plants that you can look at today and later be told that all the generating units are on the line and you can't see anything. Well, there weren't many like that fifteen years ago. And there have been a combination of motivations that have achieved this. It's not just the hardware suppliers that have done it. It's been a motivation drive on the part of the plant owner for a standard of excellence, his willingness under various pressures, his willingness to invest more money on a bigger box. As I say, these motivating forces have combined to produce some excellent achievements in improved hardware and improved operating techniques.

66. F. E. TEMPLETON--I'd just like to make one comment. I'm a little perplexed, Mr. Thompson, at your question. We just got through spending hundreds of millions of dollars on what we've got now and you're suggesting what's the next step and what are we going to replace all this stuff with? I think we ought to concentrate on what we have now that works rather reliably. The other thing is, if the problem's solved, that is, if pollution is abated to the degree that our society is agreed that it ought to be, then I really have to question how much more effort in research to improve efficiency should be put forth. Perhaps we should focus research on making the existing systems more reliable and on processes that will use less energy.

67. G. THOMPSON--Mr. Templeton, I didn't intend to challenge the industry's record in application of existing technology to control existing plants. My concern is the future---those cases that involve the installation of complete new plants, expansion of existing ones, or replacement of worn out equipment. We should be concerned with a number of factors: control efficiency, equipment reliability, maintainability, prevention of performance degradation, energy consumption, and the like. What efforts are being made to make air pollution

control more cost effective, more affordable? Are there technological improvements on our horizon which may come to have a significant impact in reducing the costs of air pollution control?

68. F. E. TEMPLETON--I think your point is pretty clear, but I do think that you have to recognize it both from a regulatory standpoint and what kind of rules should we be looking at in the future as well as what kind of technology should we have. There seems to be a tendency to want to apply any new technology to everything that's already in as soon as it comes out, and I'm not just accusing EPA of that becuase I think there're some pretty sensible things done there, but I do think a lot of environmental groups, for example, as soon as word gets out, the, "Here's a new technique or a new technology," there's pressure to apply it everywhere, even with existing plants, and often before it has been thoroughly demonstrated and proven in actual practice. Researchers sometimes forget about the need for sustained reliability. So while it may be that future plants, as they are designed, can be built with better technology from an energy standpoint, there is sure a tendency to run with the ball as soon as it pops up even with the plants that have already made substantial investments.

69. G. THOMPSON--I might add---being part of a regulatory agency ---people sometimes do tend to run with some of the research results that we produce. We have to be very cautious that what we produce is correct, is factual, and does the job. Allow me to provide one more example in the wastewater area. Some of the "lime and settle systems" I've seen in the United States are not too effective in metals removal. There are new technologies that have been developed in the past few years that don't produce a sludge. Sludge may not be one of your industry's problems, but it certainly is a problem with other industries. If we don't advance the state-of-the-art for certain existing technologies, we're not taking care of the total problem, and we obviously won't be preparing for the future.

70. S. CALVERT--I don't want to say too much because Jim Abbott is going to give a paper on new technology, and I hope he may even mention something we're doing. In general, we may not have discovered any great new principles for particle collection, but I think we have learned a lot about the way particles behave and what can be done to improve collection. If one is able to do a good engineering design job, he'll take advantage of the conditions that exist and use them to improve collection efficiency rather than perhaps fighting against it. There are many installations where the equipment simply didn't use what was available to the best advantage. Then, I'm struck with another thing in listening to

the papers here, that in many cases the question wasn't whether the scrubber could give the proper collection efficiency but whether you could just keep it on line, and whether it would be free from being plugged up by scaling or otherwise being made inoperative. So putting things together in different combinations, and different configurations that are easier to operate and easier to maintain can offer some real advances. And this is an area where people in industry with the problem have to tell the people in R&D just what the problems are so something can happen which will be beneficial.

71. L. C. TROPEA--Dick, I have one comment to pick on this water pollution commentary that was just made. Pretty soon, say sometime in the 1980's, we're going to have, for the first time, the major legislation in air, water, and hazardous wastes, all out and in final form. EPA's effluent guidelines program is restudying everything in the world. Limitation on the traditional pollutants, like the total suspended solids, pH, and things like that, are things of the past. They are going back now and looking at a whole bunch of substances which many of us would have difficulty even spelling. In the very near future, they are going to amend all these guidelines. Anybody who's got a water discharge from air pollution control systems is going to be faced with some real challenges with some of the limits that they are going to be imposing in this area. When you take the new water requirements and add the RCRA requirements, the hazardous waste requirements, which will eventually require any sludges that you produce at any waste treatment facilities or any baghouse catch that you make, there will likely be some new ground plowed in employing a system's approach to future air-quality decision-making.

72. F. R. CULHANE--George, I might just make a comment that there was a development on the horizon in the fifties that was exciting. It was a sonic agglomerator. And those of us that were doing the work in those days looked at it as an opportunity to increase the throughput in fabric filters for a given pressure drop or to reduce the energy requirements of the wet scrubber, or to reduce the size of the precipitator. The energy requirements of the sonic agglomerator proved to be unattractive and the device faded and never really became commercial. But it is something, perhaps, our industry should reinvestigate. If we eliminate the fines and have only coarse material to collect, we could reduce the size and cost of the control equipment.

73. E. L. COE--I'd like to add a little on my previous comment on combinations of effects to improve particulate collection. There's been a tendency over the years to address particulate collection in terms of one device, that is a baghouse, a precipitator, or a scrubber. I think that the nonferrous metals

industry is probably the leader in handling their pollution problems, air pollution problems that is, with a train of devices. The hot precipitator, followed by the scrubber, followed by the acid mist precipitator, followed by the sulfuric acid train, is a tremendous combination of equipment. But, it does illustrate one of the principals that is assisting in this area and that's the application of a series of devices each operating in its own best regime. The electrostatic enhancement of scrubbing, or, in terms of scrubbers that have a problem with outlet emissions, use of scrubbing followed by wet electrostatic precipitators is a way to achieve levels of control although not necessarily levels of cost that are highly attractive.

74. JOHN D. MARTIN--BHAS of South Australia-There has been no mention tonight of venturi scrubbers, and I'd like to direct a question to the panel or anybody else who's had experience with them. At Port Pirie, we use long-throat venturi scrubbers in our sinter plant circuit, and we get quite good efficiencies with these units. Our problems at the moment are concerned with the out-of-balance on our fans due to carry over of solids probably in our liquors. My question is, does anybody here have any experience to show that the long throat venturi is, in fact, more efficient than more commonly used short-throat. We put these in at the recommendation of a consultant and unfortunately haven't been able to compare results. Secondly, I wonder if this buildup which we are getting on the fan, which is giving us our out-of-balance problems, isn't connected with our inefficient operation of our cyclone and scrubber. Would someone like to comment on it, just for interest?

75. S. CALVERT--So far as the effect of the throat-length on the effect on venturi scrubber efficiency goes, we have a mathematical model to predict venturi scrubber efficiency, and it indicates that throat-length is important up to a point. There's no one number that indicates how long the throat should be but, in general, probably lengths on the order of two or three feet are adequate. For very large diameter throats, this may be larger. But there is a mathematical model, and it's presented in an EPA publication entitled, "Particulate Control Highlights, Performance, and Design Models for Scrubbers" and we have another previous report that deals particularly with the venturi scrubber model. So to sum up, there's a theoretical reason for believing that the longer throat is more efficient. To determine whether this is really so, it would be necessary to look at your collection efficiency data and compare them with other data from venturis and similar scrubbers with shorter throats. We'd be very interested in getting some data for that purpose. The problem of the fan going out of balance is very likely due to liquid entrainment being carried into the fan

and might require asking one more question. Do you have any sort of mist eliminator between the venturi scrubber and the fan?

76. J. D. MARTIN--No, we haven't. We have a cyclone separator to trap the liquor, but we have not any mist eliminator at all. That was something that I thought that we should probably try. Initially the fans were made of mild steel and we had corrosion problems and buildup. Then we changed to stainless steel, and these are giving reasonably satisfactory performance. But we are still getting buildup which requires cleaning three times a week, which is excessive.

77. S. CALVERT--Well, a cyclone separator is one type of entrainment separator or mist eliminator. They can be efficicent, that is, they can be adequate. However, it is possible to overload them by having too high an inlet velocity so there can be reentrainment of the liquid drops that are deposited on the cyclone walls. So it's a matter of looking into specifics of the case. I think this problem of solids deposition or entrainment carry-over is a very common one and really was the basis for EPA funding several research contracts which we performed to study entrainment separator characteristics and to define them. Again, we've written some reports, some EPA publications on that subject and if you'd like afterward, I can give you some references to those.

78. E. L. COE--I have a slight additional comment with regard to mist eliminators. Mist eliminators used with most scrubbers in present practice are basically mechanical collectors. They are designed to collect liquid droplets as particles. They do have a limit in terms of the energy that is put into them which limits what they can collect. If all the droplets came through large, you would presumably collect them all providing you didn't have a reentrainment problem. The problem is that nobody told the droplets that they had to stay that nice large size. For any mist eliminator of a mechanical collection type, inertial, impaction, or centrifugal (and obviously some are better than others), there does appear to be a limit as to how clean it can get the gas stream in terms of the entrained particles, particularly if a portion of the particles are of extremely small size. In this connection, I refer back to the remark I made previously about putting another device behind the scrubber to do a portion of the collection job. I'm not particularly promoting wet precipitators, but that is one way of doing it in the extreme.

79. F. E. TEMPLETON--I might just add, we've had this same problem with scrubbers and we've tried everything, such as coating the fan blades, and I guess you probably have done the same sort of thing. We have one unit that has a fan on the

hot side and the cold side in order to get enough pressure drop. The hot side fan doesn't have any problems at all. So one of the approaches we were going to take was to put both fans on the hot side so we won't have the coating problem on the other fan. But at least with us, we've had that similar experience with that type of scrubber. When it works, it works great, it really does a good job but it doesn't work very long.

80. S. R. OREM--I'd like to add just a general comment, Mr. Martin. On some of the very high efficiency commercial scrubbers in the U.S.---I think particularly of the BOF furnace venturis in the steel industry---the tail throat is not nearly the length of those in the pictures you showed us yesterday. That's a general answer. Regarding your fan buildup problem, have you thought of either steam or water cleaning the fan in service? I think of applications in the pulp and paper industry and applications in the steel industry where it has become necessary to apply a cleaning force to the fan in service. This may be something worth looking into.

81. PETER Z. CZUCHRA--FMC Corporation-In reference to the question regarding the importance of throat length on scrubbers, a few years ago we developed our own scrubber that does maintain a throat length of approximately two to three feet. We did test it against some orifice type scrubbers we have at some of our chemical installations and found that at a given pressure drop, the scrubber we developed did give a lower outlet gain loading. So, I would think that our practical experience tends to agree with Dr. Calvert's theoretical data.

82. K. SEMRAU--The first comment I would make is about the reentrainment problem with the mist eliminator. I could concur generally with what other commentators here have said in that respect. I would say that this is probably a case of reentrainment from the cyclone, which is not uncommon. The major problem with many entrainment separators is that they are undersized and they are overloaded, as Dr. Calvert has indicated. There's an obvious reason for this. The entrainment separator is usually the part of the system that costs the most money, so there is a tendency to cut corners and to keep the unit as small as possible to avoid the added cost of a larger piece of equipment. With respect to the matter of throat length and its effect on efficiency, I would harken back, essentially, to work which my colleagues and I have reported in the past. As far as we are able to see, both on the basis of our own and other people's data that have been in sufficient supply to yield to interpretation, it doesn't make any difference whether the throat is long or short or for that matter, whether it is a throat. We've

gotten essentially the same sort of performance from pipe line contactors, from orifice contactors, and venturis of different shapes and sizes.

83. R. W. LEE--I'd like to take this question of cyclonic separators a bit further. For several years now, I've been vainly trying to acquire design information on cyclonic separators, and I have acquired a certain amount just by collecting sizes of various separators. But, I've looked through quite a lot of Dr. Calvert's work, and I haven't seen very much in the way of design information about cyclonic separators. You've certainly done a lot on mist eliminators, but I would be very interested in any reference you can give on this particular subject. But going on a bit from that, after this particular problem about fan build-up, I think it depends on what efficiency you're getting from the venturi itself. If you use low pressure drop venturis on fine size fume, you don't really remove a great deal of it. And, if you add up over a day or three days, or seven days, or whatever it is, the amount of actual penetration you are getting through a scrubber like this, you'll be surprised how much solid matter the fan is actually handling. Depending on what type of fan you're using, whether it's a paddle blade or whether it's a more efficient fan on which fume tends to stick more easily anyway, you can get these problems.

84. S. CALVERT--With regard to design of cyclone separators, if you're talking about designing for the primary collection efficiency, by which I mean the efficiency with which the drops are first removed from the gas, we presented a model for that in the Scrubber Handbook which is an EPA publication and the model is essentially the Leith and Licht model for particle collection by cyclone separators. It takes into account the turbulent mixing in the cyclone. We went on from there into an experimental study of cyclone separators as mist eliminators or entrainment separators in order to investigate the reentrainment velocity which essentially limits the capacity of the entrainment separator. We had found some conflicting statements in the literature indicating that you could use anywhere from 45 ft/sec to over 100 ft/sec inlet velocity, which gives you quite a bit of room for confusion. So we did experimental work on one type of cyclone separator which actually was a design which we lifted from Nonhebel's book. I believe it's a Stairmand design. This is a cylindrical body with a top inlet with the anti-creep skirt on the outlet and an anti-swirl baffle in the bottom. We were unable to get reentrainment at inlet velocities up to something a little over 100 ft/sec. So, we have some limited data on the reentrainment characteristics of a cyclone separator used as an entrainment separator. And we have, as I said, presented design methods for computing the primary collection efficiency and indeed have

repeated it and improved it in this little report on the design methods for scrubbers.

85. R. W. LEE--Just a quick point, I've looked at these methods for calculating cyclone efficiencies. But as far as I can see, nearly all the cyclonic separators that I've seen have bottom inlets and top outlets whereas most of the work seems to have been done on the dust cyclone which is rather different. You know, there must be some significance to the fact that the inlet is on the bottom instead of the top. This must have some effect on its performance.

86. S. CALVERT--I don't think the drops know about it. And there are cyclone separators that are used as entrainment separators in which the gas inlet is at the top, and they are essentially analogous to cyclones used for dust collection. The problems that we could get into with bottom inlet and top outlet type cyclone separators are more of the type related to operations. There can be solids depositions, there can be very severe problems due to that. Where solids can be deposited and can impair the efficiency of the cyclone separator and can cause very serious maintenance problems. But so far as the primary collection efficiency goes, there's no reason to believe that it should be any different assuming one has taken all the variables into account.

87. E. L. COE--I have an additional comment. I cannot add to what Dr. Calvert has said about the design of the cyclonic separators; however, if you wish to look at more literature on the subject, a good deal of it is contained in the patent literature on the separation of droplets from steam.

88. TOM C. SUNTER--Carborundum-Some comments were made this morning about the use of an extra compartment in a dust collector for maintenance purposes and the possible use of opacity meters after each compartment to detect a broken bag. Has the panel any experience in the use of such units for detecting a broken bag in a compartment and will that find the bag? Our experience is that if you don't catch a broken bag right away, you'll have five of them or more and the sooner you do it the better job you have. Has the panel any experience?

89. E. L. COE--There is a unit at the Pennsylvania Power and Light Company that has been in service now for about five and a half years, and it has been found at that installation that the compartment in which there is a perforated bag can be located by correlation of the opacity reading on the outlet and the timing of the cleaning cycle. This is very convenient at this particular station because the opacity meter and the cleaning cycle control panel happen to be side by side. The indication of a failed or perforated bag appears when the

compartment having the perforation is on the cleaning cycle the opacity drops because that leaking compartment is then off line. This is a reverse air baghouse. It has been found, as recounted to me by the station personnel, possible to locate a perforation equivalent to a 1/4-in. diameter hole in one bag in one compartment. For reference purposes the baghouse has approximately 5000 bags, and I do not recall the exact number of compartments, it's on the order of 16, I believe.

90. F. R. CULHANE--I'd just like to support that statement that Lee has made. We have an opacity meter on a large 2 million cfm baghouse. The opacity meter picks up a blip, it comes on when a leaking compartment goes off the line and is no longer filtering. You can't pick it up with your eye in the stack, the stack still remains clear until we're down in the order of magnitude maybe .004, .005 or something like that. So the opacity detection system on this installation is extremely sensitive. Prior to opacity meters, we used to be able to do this visually. In a carbon black plant you could see a small plume coming off a stack, and when a leaking compartment would go off for cleaning, the stack would clear up. Then you knew that you could isolate your problem in a compartment. But the opacity meters on the market today are much more sophisticated and much more accurate than the naked eye. It's a very practical approach.

91. E. L. COE--I would add a caution, however, (1) you have to have a good opacity meter well maintained or you are going to miss the effect, (2) you need to have a good correlation between the opacity readings and the cleaning cycle. If you do not have the opacity meter at the panel that indicates the compartment being cleaned, you would need to, somehow or other, log the two operations so that you could correlate them and (3) if you get several such compartments simultaneously, you never will be able to find out which one it is. Therefore, you need to catch them as they go.

92. E. S. GODSEY--The latest baghouse installation that we have installed has a system very similar to the one described except it carries it a little further. We have an opacity meter that monitors the total output for the baghouse and this one is rated for about 200,000 acfm. The instrumentation is arranged so that if the opacity reaches a pre-set limit, the baghouse will automatically go into a shut-damper, delay, open damper, and then continue on through the 16 compartments until the damper that is closed causes a reduction in the opacity reading. If that happens, the damper will automatically stay closed, the compartment is taken off the line until the operators can get around to going in and finding the broken bag. If the first compartment that has a broken bag in it shows a decrease in opacity, but not to the

set point, that damper will stay shut, the cycle will continue until it reaches the second compartment that has a broken bag and in that case we'd have two compartments out of service. So, it's a system similar to the one described except that it's automatic, and it does locate by actively seeking out the bag compartment that's causing the problem. The instrumentation costs almost as much as the baghouse.

93. GEORGE KAZONICH--Phelps Dodge, Morenci, Arizona-Getting back to Mr. Billings' original comment on the precipitator efficiency versus maintenance, I'm one of the people who crawls into precipitators and cleans them out and keeps them running, and I think we had some of the highest and some of the lowest points on his testing. And it comes down to the point of how much effort are you going to expend to keep up with that 99% efficiency? One of the highest points that he recorded was after a two months rebuild of a unit with total plate straightening, replacement of outlet plates, replacement of virtually all wires in the outlet section, tuning up the electronics by the manufacturer's representative, just on and on and on through the entire unit. But obviously you can't go through a two-month rebuild of the unit, put it on line for a week, take if off and do the same thing again. There has to be some economics put into this thing to say, well, we can maintain a 90% capture or a 95% capture, or 98% efficiency, whatever. But economics have to get into this somewhere or other. Because you can't just clean a precipitator every day in order to keep it up at 99.9% all the time. And one other comment, I'd like to side with Mr. Billings on one thing, because he's requesting some guidance from EPA and I know Arizona has been spanked on the hands several times because the guidelines that they've come up with or their presentations for control parameters are generally declined as being unsuitable by EPA. So, I think the fact that he is seeking guidance here is showing forethought on his part and that it hasn't seemed to come from the EPA people at this meeting is a bit sad.

94. R. L. MEEK--If I may comment on your question, Mr. Kazonich--The purpose of this forum is an interchange of technology and information, not to respond or supply guidance from EPA which should be available through other channels. I think the answer to the first part of your question on the operation and maintenance of precipitators probably addresses several members of the panel. Let's start with Dr. Templeton.

95. F. E. TEMPLETON--I think I would have to agree with the idea that you certainly have to bring economics into the picture on any piece of equipment, not just precipitators. Our experience with our precipitator is perhaps a little different than what Mr. Billings had in mind because our precipitator is not on the terminal end of the control string. It's

on the front end of it and so there are other pieces of equipment after the precipitator that take other materials out. We haven't had any particular problems with our precipitator from a maintenance standpoint. Ours is designed with a spare capacity in terms of the number of sections to it and so we can take a section out and still run the precipitator and do maintenance work. That's one approach that is useful in keeping it on line. I'd go back to a point I made earlier, that is if you are going to choose a specification for emissions limitations for a plant, you have to look at that particular plant and I think that you will find that that specification really should not be a single number but a number that reflects the operation of the plant itself, the process, as well as the fluctuations and efficiency of the equipment, that takes into account the changes that take place over time and the efficiency because of the degradation of the collection efficiency over time. Again, I don't think it's correct to sample when you just finished doing a cleaning job on the precipitator. I think you need to take enough samples over a long enough period of time that the overall performance is evaluated and can be looked at in light of what it really does, not just what it does in ideal conditions which never really exists except for a particular instance of time.

96. E. L. COE--One of the questions that is being raised here, at least by implication, is the performance of control equipment over time, and one of the problems in accessing this factor is the fact that we have only a cumbersome, relatively instantaneous method of measuring mass emissions, yet we are characterizing everything that we are talking about in terms of these mass emissions. And my comment is this, that it would behoove everyone who is working on measurement devices, Bruce Harris included, to look for a way of instantaneously or continuously measuring mass emissions over time.

97. C. H. BILLINGS--I'd like to respond to some of the things that were said. In the first place, I didn't bring the question up to seek guidance and go right home to Arizona and write a regulation and that would be it. We do get along pretty well with EPA although we have our differences; I think all the states do. I didn't want to cause any dissention in the ranks as it were. The reason I brought the question up is that I thought it would be an interesting discussion, and it was germane to what we were talking about tonight. Some of the things that we have to consider on any dust collection device in determining what the efficiency is and should be are based on certain theoretical precepts and are also based on certain practical precepts. For instance, in electrostatic precipitators dust buildup on the electrodes and configuration of the electrodes, the wires,

reentrainment, resistivity, that sort of thing, once your precipitator is built and in place, those are fixed parameters. There isn't much you can do about them. It can be quite expensive, in fact, to correct some of them. The practical aspects of this are that you might have poor maintenance, or inadequate voltage control, improper conditioning of the material going in, etc. So I think we have to consider both the practical and the theoretical aspects of any dust collection device. What I was seeking tonight was to get some guidance as to what the electrostatic precipitator on a copper plant was capable of. I didn't exactly want one number but I just wanted to know some sort of guidance as to what such a device is capable of. As I said before, I'm sure it's better than 74 and a half.

98. R. L. MEEK--Any other comments?

99. D. B. HARRIS--Dick, could I just get in a commercial while I'm here? Since everybody has been participating so well in the symposium, I just wanted to mention a couple that we sponsor that might be of interest to you on the measurement of particles. One that will be coming up in October in Daytona Beach, Florida on October 7th through the 10th is titled "Advances in Particle Measurement and Sampling" and a call for papers is going out. If anybody has been doing any unique efforts in this area, we'd be glad to hear from you. We had a very successful one last May in Asheville, N.C. The proceedings for that are now available if anyone was not able to attend and would like to receive a copy of that, get in touch with me back at IERL or with our reports distribution center. We also sponsored one on fugitive emissions, the last one was in October in San Francisco and hopefully we'll have the proceedings of that soon. We're planning on May of 1980 as being the time of the next meeting on that, but we have yet to pick a site.

100. R. L. MEEK--No further questions? Well, I think we owe the panel a round of applause for their participation. And gentlemen, I thank you. This session is adjourned.

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